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JUNE 1950

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FEATURES

ARTICLES

TRANSACTIONS

Cover photo: Trapezes, or suspended platforms, used at the St. Joseph Lead Co. in wrapping mine pillars with old hoist rope. The traps are used on pillars over 35 ft. high, which height exceeds the range of sectional ladders.

Correction on last month's cover: We erred in spelling the photographer's name. Correct spelling . . . Frank Day.

JOY L-111 Hoist, powered by JOY's reversible "Pistonair" Motor

JOY SINGLE DRUM HOISTS

More for your money in every way!

- ★ **More Compact**
- ★ **Lighter Weight**
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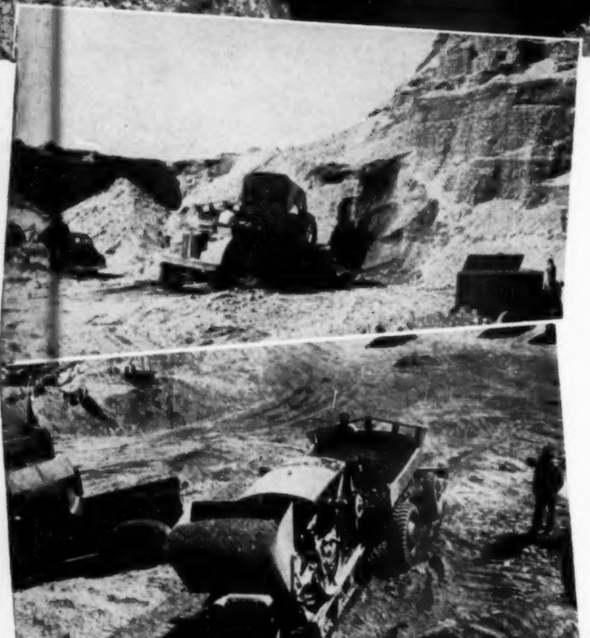

Eimco 104 RockerShovel loading sand and gravel (weight: 125 lbs. per cu. ft.) at the rate of 8 yards per minute.

"A crawler-mounted RockerShovel for rock." This is the cry we've heard from RockerShovel users for many years—and now, after years of field testing, we have the machine our friends have asked for.

The 104 Eimco RockerShovel is the same rugged type of machine that has been the preferred loading equipment underground for many years—the same overhead principle pioneered by Eimco — the same heavy-duty construction throughout.

The Eimco 104 RockerShovel is designed for loading heavy, abrasive rock or sand and gravel using a 1¼ yard bucket for heavy rock, and a 2 yard bucket for sand and gravel. The 104 is available with a 48 H.P. Diesel engine, or 30 H.P. Electric motor.

This machine is ideal for loading, bulldozing, or cleaning in roadwork, quarries, stock piles, and mines. Operators are finding new uses for it every day.

Write for more information.

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Letters to the Editor

PROFESSOR TAGGART'S article on Engineering Education in MINING ENGINEERING for April is very much to the point. I agree with his objective completely, and wish to congratulate him on stating so clearly what many of us feel but are not sufficiently articulate to express.

However, it is probably necessary to educate the professors before the benefits can be passed on to the students. Under our educational setup a man reaches a responsible teaching position in engineering by virtue of being a specialist; he naturally believes in the importance of his specialty, and passes it on to his students, who lack the maturity to get a proper perspective. As a result, the students' engineering education consists of separated packages of knowledge, each of which may be over or under developed according to the personality and forcefulness of his instructor in that subject.

I feel that the reformation must be instituted at the top, with the selection and training of the teaching staff. Wide experience, a general knowledge of science, an immense desire to coordinate and apply all possible scientific principles, and an increased respect for students as human beings, should be requisites for teaching.

This change in attitude can best be accomplished by emphasizing in professorial circles the desirability of *unifying and coordinating* scientific principles. Only after the idea is accepted that a generalized training is superior to specialized and departmentalized knowledge, can any headway be made in giving students the kind of preparation they need. Until it is accepted the few teachers who try to give their students a generalized training will be suspected of encroaching on the territory of others.

Perhaps a senior course in "Engineering Applications" would help. It should be taught by an engineer of wide experience and imagination who is not a specialist, and only problems involving several engineering subjects should be considered. Such a course might reduce the many years of experience required before graduates reach full usefulness.

Knowledge in a dictionary or encyclopedia is listed alphabetically, with plenty of space for inserting new material as it develops, and items are not omitted merely because they cannot be classified under a particular subject. However, this is not true in engineering where our list of subject titles is limited to well developed sciences. Engineering, like Herbert Spencer's conception of evolution, is *successive differentiation and integration*, and it may be that we have learned differentiation well, but are only just beginning to appreciate the more difficult subject of integration.

Fred C. Bond,
Technical Director Basic Industries Research, Allis-Chalmers Mfg. Co.

We Stand Corrected

We have read with interest an item contained in your January 1950 issue under the heading "Barite", (p. 70) in which is mentioned that a company, Maritime Barytes, Ltd., opened newly developed deposits of barytes ore in Nova Scotia. The article infers that production from this deposit is responsible for competition with American producing mines.

We should like to bring to your attention that our company, with its mine and plant situated at Walton, Nova Scotia, is the only barytes producing company in Nova Scotia, and we know for a certainty that the referred to deposits of Maritime Barytes, Ltd., are at no stage of production.

Canadian Industrial Minerals Limited
W. W. McBrien, Secretary-Treasurer.

Comments on "Cerro Bolivar"

. . . Mr. Lippert writes in his article [Cerro Bolivar—Saga of an Iron Ore Crisis Averted, MINING ENGINEERING, Feb.] that: "In general, Cuban laterites are not competitive with other ores, and persistent surveys by several companies have never uncovered any high-grade iron deposits of sufficient size to warrant interest as a dependable source of supply. Small deposits occur along the west coast of Pinar del Rio Province containing 1 million tons of open pit ore. . . ."

The information concerning the iron resources of Cuba is unfortunately incomplete, inasmuch as there is a deposit of Bessemer-grade ore in the southern coast of Oriente Province, where more than 300 million tons can be proved by simple exploration. Notwithstanding the existence of informative documents which detail the facts about this ore bed, a field inspection can be made by experts at small cost, as transportation facilities are readily available.

This deposit, entirely within 15 miles of the coast, can be conditioned for operation for one million dollars, a striking contrast with $50 million for El Pao and $113 million for Cerro Bolivar. Considering the relative distances by sea, from the southeastern part of Cuba and from Venezuela, to either Mobile or Baltimore, the saving in the cost of marine transportation is quite evident.

Furthermore, in case of war, the sea-route from Cuba could be more easily and effectively protected. . . .

Rodrigo Rodriguez E.
Havana, Cuba

Authors in This Issue

Paul M. Tyler (p 694) is a leading mineral technologist and economic consultant, with 32 years in government work behind him. Also, "a good many hundred" government publications, professional society papers and trade journal articles have passed through his typewriter. Following his graduation from MIT in 1912, he had "the usual knock-about mining and smelter experience" and did some ore-testing research work under the noted Prof. Charles Locke at MIT. He joined the government during World War I, working

P. M. Tyler

J. L. Gillson

with the Tariff Commission, and later with the USBM, as Eastern regional engineer. During World War II he headed wartime prospecting and laboratory investigations in 21 eastern states for the Bureau of Mines. He's no stranger to Europe, did investigations there in 1919-20, and returned last summer for a 24,000 mile jaunt through Western Europe, Greece, Turkey and French Africa as an adviser to ECA. Mr. Tyler has also found time to write a book, "From the Ground Up," containing basic facts and figures on the American mineral industries. He has been an AIME member since 1916. In his leisure hours he enjoys gardening, swimming, and fishing.

Joseph L. Gillson (p 685), has demonstrated a long and continuing interest in nonmetallics, and it is this interest that has taken him around the world twice, and that made him Chairman of our Industrial Minerals Division in 1948. Since 1928 he has been a geologist for the E. I. du Pont de Nemours Co. in Wilmington, Del. Dr. Gillson attended Northwestern, and then it was "time out" for service as commissary officer on a transport during World War I. He returned to take a Ph.D. at MIT, assisted in mineralogy at Harvard, and then went to teach mineralogy and petrography at MIT for five years before joining du Pont. His travels since then have been partly reflected in the two TP's he's presented before the Institute—on fluorspar deposits in the western states and in Spain. Now, in this issue, he takes us to Brazil. Dr. Gillson's AIME membership dates from 1923.

M. D. Cooper (p 718) is director of mining engineering education for the National Coal Association in Pittsburgh. Before joining the Association in 1946 he was division superintendent for Hillman Coal and Coke Co. The early years of his career were spent as an engineer for Ellsworth Collieries, and then as safety engineer for Ford Collieries. Mr. Cooper was born in Buffalo, N. Y., attended high school there and went to Philips Academy, Andover, Mass. He took his Ph. B. from Yale in 1910, and his E.M. two years later. AIME member Cooper also serves as Chairman of our Student Relations Committee. Aside from his interest in the young men of the industry, Mr. Cooper takes pleasure in stamp and mineral collecting.

W. L. Hill (p 699) co-author with Messrs. Armiger and Gooch, is a chemist with the Dep't of Agriculture and lives in Washington. He has been a professor of chemistry at Milligan College, and before that an instructor and principal at Etowah High School, Etowah, Tenn. Born in Chilhowie, Va., he attended high school there, and then went to Milligan College, the Univ. of Tennessee, and the Univ. of Virginia, taking B.S. and M.S. degrees in chemistry.

C. H. Snyder (p 715) is president of the Sunnyhill Coal Co., in Pittsburgh, which developed the Colmol continuous mining machine. Born in Pittsburgh, he attended Dormont, Pa., High School, and immediately after graduation the enterprising Mr. Snyder formed the partnership of Snyder and Swanson, coal brokers. He has been secretary of that company since its formation in 1931, and president of Sunnyhill since its formation in 1939. For relaxation, Mr. Snyder retires to his 500 acre dairy farm in Washington County, Pa.

C. H. Snyder

J. R. Thompson

J. R. Thompson (p 658) is a graduate of the Georgia School of Technology, with a B.S. in textile engineering. He has been with the Atlanta and Denver districts of the industrial products sales division of B. F. Goodrich, and also at their Martha Mills in the textile division. He hails from Ashtabula, Ohio.

D. E. Norquist (p 712) was born in Kansas City, attended Southwest H. S. there, and Wentworth Military Academy in Lexington, Mo. He has also studied metallurgy at the University of Kansas City. Since 1934 he has been associated with the Sheffield Steel Corp. in his native city, as secretary and assistant to chief metallurgist, chief metallographer, and as a field metallurgist and trouble shooter on armor piercing and high explosive shot steel during the recent war. He is now manager of the grinding media division there. AIME member Norquist turns from grinding to golfing in his spare time.

(*Continued 2nd col. page* 648)

Engineering Societies Personnel Service

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18**; 100 Farnsworth Ave., **Detroit**; 57 Post St., **San Francisco**; 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6¢ in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter or $12 a year.

Positions Open

Mine Foreman, technical graduate, about 35. Must be experienced in low cost mining operations and in shrinkage stoppage. Location, East. Y3593.

Engineers. (a) Assistant Geologist for underground and surface work, largely geological mapping and structural interpretation. Broad experience not necessary; young graduate considered. Salary, $3000-$4200 a year. (b) Shift Bosses, Latin American experience preferred; knowledge of Spanish essential. $3300 a year. Location, Colombia, S.A. Y3587.

Design Engineers, with considerable experience in mining plant and machinery design. Salary open. Location, northern Minnesota. Y3583.

Placer Engineer, 35-50, to assist chief engineer on development gold placer fields. Prefer man experienced in dragline washing plant operations, underground mining or dry stream beds, diversion of streams, construction of reservoirs, flumes and pipe lines. Three year contract. $7000-$8500 a year, tax exempt. Location, Ethiopia, temperate climate, elevation 5000 feet. Y3556S.

Assayer capable of doing fire assays on gold and silver ores, bullion assays, lead and zinc wet assays, and possibly some qualitative and quantitative analysis. Knowledge of Spanish desirable. Some experience desirable. Salary, plus bonus and vacation. Location, South America. Y3410.

Junior Metallurgist, graduate in ore dressing, some experience in ore dressing laboratory test work. Standard three year contract. $3000 a year plus bonus one month yearly. Single status. Knowledge of Spanish desirable. Free transportation to Bolivia by air, four weeks vacation yearly, free living quarters. Location, Bolivia. Y3334.

Engineers. (a) Metallurgist-Mill Foreman, graduate, experienced ore dresser and mill operator. Working knowledge of Spanish essential. Standard three year contract. $5000 a year plus bonus. Single status for six months. Free transportation by air for employee and wife. Four weeks vacation yearly, free living quarters. (b) Mine Foreman, graduate, experienced metal miner. Working knowledge of Spanish essential. Three year contract. Salary, $4200 a year plus bonus. Single status for six months. Free air transportation for employee and wife. Four weeks vacation yearly, free living quarters. Location, Bolivia. Y3311.

Junior Geologist, some experience in mining geology, for large mining company in Bolivia. Three year standard contract. Knowledge of Spanish. $3600 for first eighteen months; $4200 thereafter. Y3305.

Men Available

Mining Engineer, 31, married. B.S. Min. Eng. Seven years' experience non-metallics; glass making materials, fluorspar, feldspar and others; experience from operational and research viewpoints concerning exploration, mining and processing. Reports. Desires position research or exploration leading to new development and as eventual plant manager or engineer. Available 30 days. M-544.

Operations Manager, graduate mining engineer, 35, married, 2 children. Seven years' non-metallics supervision and management, three years' metal mining, two years' metallurgy, two years' Navy. A.B., B.S., E.M., Columbia University. Property evaluation as well as managerial experience. Now located West Coast. References. Available June first. Interview preferred if practicable. M-545.

Authors *(continued)*

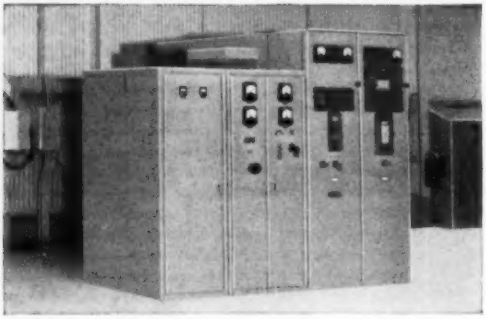
C. M. Marquardt (p 703), a native of Gaylord, Mich., attended the Michigan College of Mining and Technology, taking a B. S. in mining engineering and an M.S. in geophysics there. Beginning as assistant manager of the Mutual Coal Co., Gallup, N. M., he later became a consulting geophysicist for Bethlehem Steel, then mine superintendent for the Rustless Mining Co., Willows, Calif., and from 1943-44 was mine manager at the New Park Mining Co., Keetley, Utah. He is now an electronic engineer for the Combined Metals Reduction Co. in Salt Lake City. An AIME member, Mr. Marquardt has presented two other TP'S before the Institute. He likes to relax with rod and gun.

J. R. Guard has served as an electrical and mechanical engineer with the Rochester and Pittsburgh Coal Co., Indiana, Pa., for the past 19 years. Prior to 1931 he was associated with the General Electric Co. in Schenectady and Pittsburgh. Mr. Guard was born in Norwich, N. Y., and attended high school there. He took his electrical engineering degree from Syracuse University. His outside activities include golf and community activities.

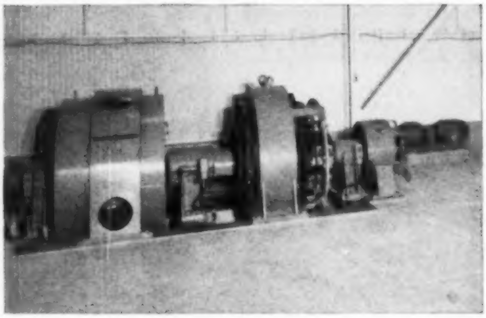
Elmer A. Jones (p 679), Assistant General Mine Sup't for the St. Joseph Lead Co., Bonne Terre, Mo., has been with that company since 1926. He was born in Minneapolis, where he attended Central High School, and he has an E.M. from the Univ. of Minnesota.

AUTOMATIC HOISTING

for continuous tonnage!



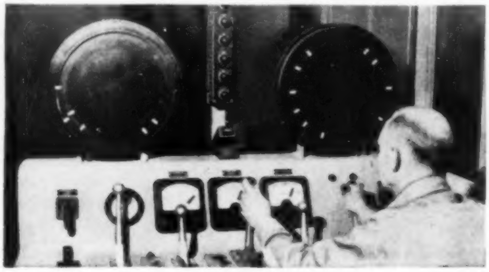
Completely metal-enclosed for personnel safety, this control cubicle houses the synchronous motor switchgear panel, plus Ward Leonard type d-c control equipment. Less space requirement and simplified control result from use of the G-E amplidyne, which eliminates need for many contactors and relays, while permitting the retention of safe limits on the hoist's acceleration and deceleration.



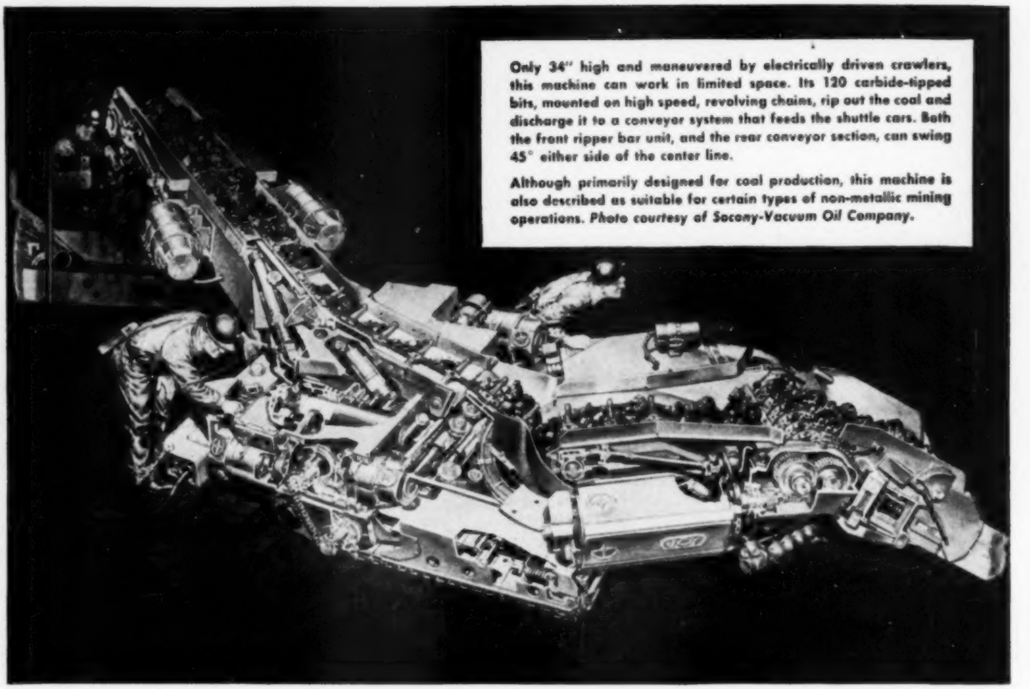
Converting a-c to d-c, this G-E motor-generator set comprises a 750-kw generator, an 1100-hp synchronous motor, and a 20-kw exciter. The generator supplies adjustable-voltage power for the operation of the two hoist motors. At right, the G-E amplidyne, used as an exciter for this generator, provides superior operation and more positive limits of acceleration and deceleration, for greater safety, increased production.

MINE-HOIST DRIVES

—to cut mining costs!



When necessary for inspection, testing, or man-trips, control of the hoist can quickly be switched from automatic to the manually operated master controller as above. Adequate starting, stopping, and maneuvering of the skips with ease, precision, and safety is provided by the optional manual operation. With the normal automatic operation, the duties of the operator may be chiefly of another nature.



Only 34" high and maneuvered by electrically driven crawlers, this machine can work in limited space. Its 120 carbide-tipped bits, mounted on high speed, revolving chains, rip out the coal and discharge it to a conveyor system that feeds the shuttle cars. Both the front ripper bar unit, and the rear conveyor section, can swing 45° either side of the center line.

Although primarily designed for coal production, this machine is also described as suitable for certain types of non-metallic mining operations. *Photo courtesy of Socony-Vacuum Oil Company.*

Here's a way to mine

2 Tons of Coal Per Minute

...with the help of NICKEL ALLOY STEELS

THIS Joy Continuous Miner can rip coal from seam faces and load it into transport equipment for removal to the surface, *at a rate of two tons per minute...*

It eliminates blasting, increases safety, reduces timbering requirements and combines operations usually done by blasting, cutting, drilling and loading.

To assure long, trouble-free performance... under conditions involving severe shocks, stresses and wear ... the builder of this electrically operated machine, Joy Manufacturing Company, Franklin, Pa., specifies nickel alloy steels for some thirty vital components.

These nickel alloyed steels provide extra strength, excellent impact values and fatigue resistance, along with resistance to wear. Nickel alloy steels may help to improve the performance of your equipment. Send us details of your problems for our suggestions. Write us today.

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Over the years, International Nickel has accumulated a fund of useful information on the properties, treatment, fabrication and performance of engineering alloy steels, stainless steels, cast irons, brasses, bronzes, nickel silver, cupro-nickel and other alloys containing nickel. This information is yours for the asking. Write for "List A" of available publications.

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N.Y.

* "This country eventually may have to rely on foreign sources for some metals, not because it does not have them here but because it may have difficulty getting the labor to mine them. Few people realize the extent to which the 'old timers' are still the backlog of the important districts." --Fred Searls, Jr.

* A new filter has been developed by Arthur D. Little, Inc., for the AEC which will remove all dust from the air. Submicroscopic asbestos fibers are used as filter media and are claimed to stop all dust and pollen while passing suitable quantities of air.

* Out in front public relationswise is the Peabody Coal Co. which has launched a 15-min television news program on WNBQ at 10:30 C.S.T., Tuesday and Thursday nights. Clifton Utley, news commentator, will tell how coal is mined today and indicate, with pictures, the improvement in efficiency and safety which has resulted from today's mechanized operations.

* Two hundred coal mining companies are using roof bolts as a means of supporting about 14 million sq ft of roof surface, Edward Thomas of the Bureau of Mines reported. Various types of metal and wooden pins can be used. The method was first used by the St. Joseph Lead Co. over 20 years ago, but is now being studied and adopted widely in metal, nonmetallic, and coal mines.

* The first ten technical graduates have reported at Hanford Works in the state of Washington, to begin the newly installed General Electric rotational training program, believed to be the first of its kind at any U.S. atomic energy plant. The program is to provide suitably trained personnel for the nation's atomic energy program.

* The need for top-notch scientists and engineers to stimulate mining programs in British colonies is acute. About 60 American experts are being sought for the ECA sponsored program. Positions for mining engineers, geologists, assayers, topographers, and petrologists are open.

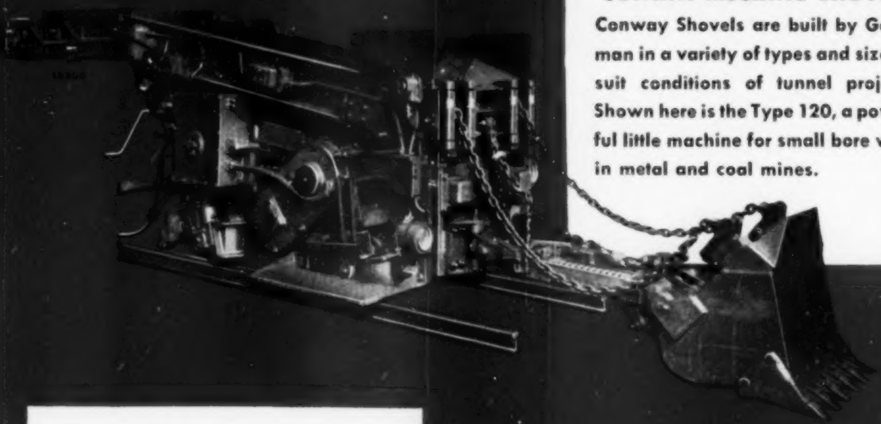
* Production of primary aluminum for the first quarter of 1950 was 322,425,008 lb which is 6 1/2 million lb more than was produced in the first quarter of 1949. Shipments of aluminum sheet, plate, and strip amounted to 259,772,157 lb for the first quarter.

* Policy on manganese procurement for the stockpile is being studied by a House Armed Services Subcommittee with the possibility of lowering specifications. The U. S. Bureau of Mines is drafting a bill to encourage domestic production for stockpiling through producer subsidies. It will have to be cleared by the Bureau of the Budget and other agencies before going to Congress.

* Oil men consider dry holes a necessary evil in their search for oil. Last year 34 out of every 100 wells drilled were dry holes. Yet Texas Technological College is attempting to drill a dry hole on its campus at Lubbock, taking every precaution to make sure it is a "duster" so its petroleum engineering students can play around with experiments without being bothered by any messy crude oil. --Natural Resources Notes.

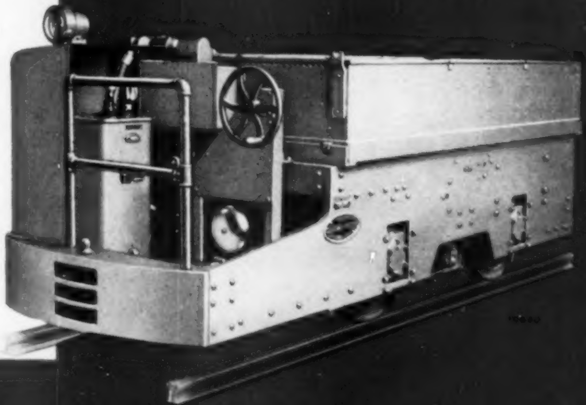
* A portable alertness indicator which signals lethargy in persons doing monotonous jobs has been developed for the Navy's Special Devices Center. A possible application to hoistmen is indicated but broaching the matter to them would be tricky.

SPEED and CAPACITY for TUNNEL WORK



CONWAY MUCKING SHOVELS

Conway Shovels are built by Goodman in a variety of types and sizes to suit conditions of tunnel projects. Shown here is the Type 120, a powerful little machine for small bore work in metal and coal mines.



GOODMAN LOCOMOTIVES

Fast hauling of heavy loads is provided for Conways by Goodman locomotives. They are available in trolley, storage battery or combination types. The storage battery unit illustrated here is in the 10 ton class and is powered by two 45 hp motors for operation at 110 volts.

Send for descriptive bulletin CL-49

HALSTED ST. AT 48TH — GOODMAN MANUFACTURING COMPANY — CHICAGO 9, ILLINOIS

Coal—A Healthy "Corpse"

DURING the past month coal industry spokesmen have been kept busy refuting the recent characterization of the coal industry as "sick." Certainly such an epithet is an exaggeration of the present situation. The industry is facing increased production costs and smaller markets but is still very big business.

Costs have gone up because of the increased wage and welfare expenditures demanded by the United Mine Workers. Wages are the highest of any industrial wage scale and welfare payments per ton of coal are approximately 4 times the average profits per ton for the last 10 years for which figures are available—1938 to 1947.

Some estimate the loss in market to oil imports to be between 40 and 50 million tons, which is a good month's production for the entire industry. This oil is being priced on a parity with coal in Eastern seaboard markets. Natural gas is also making inroads on the market for coal. The convenience, the cost per BTU, and availability (strikes have placed coal at a disadvantage) have all had their depressing effect on the coal market.

Despite these adverse conditions, the industry is strong and well able to take care of itself without government intervention. On the market side of the picture, if the proposed moratorium on oil well drilling becomes effective and more states restrict production, both of which are likely, oil prices will be higher. Natural gas has disadvantages in a high cost of providing surge capacity. Improvements in quality of coal are taking place rapidly and increased mechanization may make price reductions possible.

The industry has become more cohesive because of John L. Lewis and is now cooperatively sponsoring research to reduce mining costs and increase coal utilization. A supply of engineers is being encouraged by scholarships and close attention of the operators to the needs of students and engineering schools. The industry has recognized the value of winning the respect of the coal community by mutual aid in company-community problems. All of these programs are having a favorable effect on the economy of coal.

Some high-cost operators will go out of business but industry production will be well above prewar years. Coal operators can solve their immediate problems without the Government taking over the industry. However, Mr. Lewis will have to be restrained, as his present tactics are creating an overabundance of mines by his stop-and-go methods. His stoppages also force the use of petroleum which, the coal industry argues, should be conserved for transportation uses. He is depriving miners, consumers, and operators of their constitutional rights. If the administration must exercise some jurisdiction over the coal industry let it subject the UMW to the same controls it already extends to the operators.

Say 'Goodbye' to Nose Brick Troubles with

Sketch shows arrangement of movable firing hood with air-cooled kiln end.

NO SHUTDOWNS for end brick replacement with this Allis-Chalmers air-cooled kiln end. The segmental alloy steel retaining ring, shown above, forms a channel through which a blast of cooling air is forced.

Result: No kiln warpage; refractory brick lasts as long at discharge end as at any point along the kiln.

This one feature alone will, during the working life of the kiln, pay for itself several times over in reduced downtime and increased production... in lower refractory costs and longer life of the kiln itself.

Air-cooled kiln end design makes possible the use of a *more positive* air seal, resulting in fuel savings.

Besides the air-cooled discharge end, Allis-Chalmers rotary kilns are designed and built with a number of other important features:

- Extra thick shell plate at discharge end and under the riding rings.
- Centralized instrument control of the entire operation located on single control panel.
- Modern heat recuperation equipment.
- Constant delivery feeders and all auxiliary rotary kiln equipment.

Allis-Chalmers has built hundreds of rotary kilns . . . offers over 50 years' experience in kiln engineering. The A-C representative in your area will put you in touch with these facilities. Call him, or write for **Bulletin 07B6368.** A-2997

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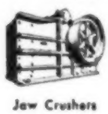
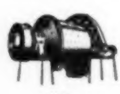
Motors


Controls


Texrope Drives


Vibrating Screens


Jaw Crushers


Mills


Gyratory Crushers

Texrope is an Allis-Chalmers trademark.

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FL2 Air Feed Leg Mounting on S48 Sinker

wherever he goes there's SPEED with the

NEW GARDNER-DENVER FL2 AIR FEED LEG MOUNTING

Small headings — sub-levels — hard-to-reach stopes — can now be drilled faster — with the new Gardner-Denver FL2 Air Feed Leg Mounting for sinker drills. Weighing only 42 pounds* — the FL2 is easily carried into remote workings — gives your runners these time-saving advantages:

- —**faster setup**—easily and quickly attached to the drill.
- —**easy collaring**—with stoper-type controls.
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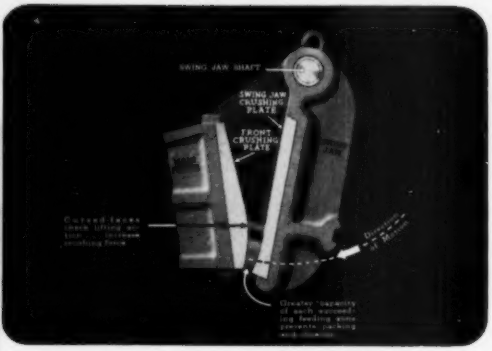

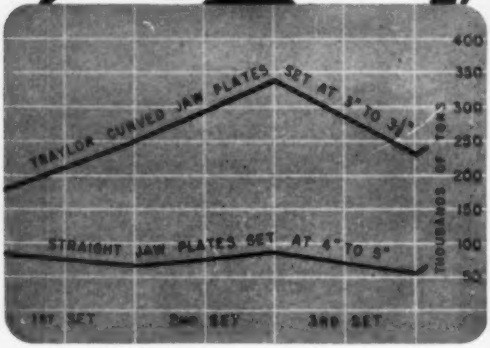

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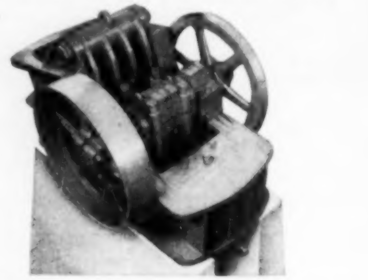

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It's Everyone's Business

MAY 17—The last bit of verbal sod had hardly come to rest on the grave of the coal industry—which grave was being eagerly dug with typewriters and microphones by administration hangers-on and even an operator or two—when its leaders gathered for the 1950 Coal Convention of the AMC in Cincinnati on April 24. Surprisingly, mourning clothes were not being worn but the boys had on their diggers preparatory to digging their way out from under the muck. High costs created by wage and welfare demands, together with work stoppages, have caused markets to shrink, but even so, an optimistic air still prevailed as new techniques were disclosed at technical sessions. The administration, however, came in for its share of knocks.

A. C. Spurr, president of the Monongahela Power Co., Fairmont, W. Va., took part of his allotted time for talking on public relations to call upon the coal industry to join forces with the electric power industry to fight any move toward socialization. He told the coal operators that the coal and the electric light and power industries have much in common since coal depends upon electricity and the electrical industry depends on coal as its principal and most economic fuel. He said both industries have been the victims of politics. He pointed out that the Federal Government is in the power business in competition with the electric power industry and stated that for every kilowatt born of electrical energy that is produced by Government water power, one less pound of coal will be used in the production of electricity by steam.

"Last year," he explained, and we quote: "the Tennessee Valley Authority was selling its water-generated power to industry at a rate of about 3½ mills per kwhr. But the private electric industry was paying 3¾ mills per kwhr for taxes alone. Our taxes cost us more than TVA was charging for its power. That doesn't make economic sense . . . the bureaucrats have been able to take electricity—which represents less than one pct of the average household budget and less than two pct of the average cost of manufacturing—and turn it into a political issue."

Mr. Spurr returned to the subject to say that more public relations work in mining communities could be used to inform the public of the danger of socialism which tends to undermine the foundation on which our success as a nation is built.

The House Public Works Committee last month conducted public hearings on the St. Lawrence Seaway and power project, with Administration officials strongly endorsing the seaway proposal. Speaking for the administration, Secretary of Commerce Charles Sawyer, Secretary of State Dean Acheson, and Army Secretary Frank Pace claimed, respectively, that it would be self-liquidating with an annual toll revenue of $36 million, would expand the economies of the United States and Canada, and would have a beneficial effect on relations between the two countries, and was vital for national defense. From industry George M. Humphrey, M. A. Hanna, and Norman W. Foy, Republic Steel, were on hand to point out the advantages of the seaway for bringing Labrador iron ore for the U. S. steel industry. Committee Chairman Whittington, Mississippi democrat, is an opponent of the seaway and says that his committee will not take action until the Senate does.

Mining lessees are among those defined as employees for social security purposes by H.R. 6000 and agreed to by the Senate Finance Committee. This bill will bring an additional 7 million persons under the compulsory coverage of the old age and survivors' insurance system, and will make available on a voluntary basis coverage for 1.5 million State and local government employees not now covered by any retirement system. Contract loggers, door-to-door salesmen, and insurance salesmen will also be covered.

Two new divisions of AIME will make their debut this fall on the calendar of coming meetings. The first of these to roll around is a bare two months away. On Sept. 1, following the American Mining Congress Metal Mining Exposition, the Minerals Beneficiation Division of AIME will sponsor one day of grinding sessions. A luncheon will also be held by the Division on Friday. In addition, the famous Scotch breakfast is tentatively scheduled for earlier in the week. Three papers presently scheduled for the meeting are D. P. Hale on center discharge rod mills, D. H. Gieskieng on capacity of single toggle crushers, and J. F. Myers on rod mill capacities and efficiencies.

Down in Washington on Oct. 9 to 13, the Washington, D. C., Section and the Mineral Economics Division of AIME will join forces for a week of meetings on subjects of national importance and in which the mining industry plays a vital role. This Division has been petitioning the Board to change the bylaws to permit the Institute to take part in such controversial issues as affect the welfare of the nation or the professional mining man. At the April Board meeting the general feeling was that the Division should carry on with unrestricted discussion and publication of such topics. This does not constitute endorsement by AIME of the opinions expressed. However, the petition was referred to a committee which was directed to prepare a liberalized version. It is to be hoped that the final arrangements will permit free discussion in the magazines and at the meetings and that should an occasion arise when the membership wished to express a concerted opinion for the record, the interpretation of the bylaws will be liberal enough to permit this.

Other fall meetings of the Mining Branch include the Fuels Conference, which is a joint Coal Division, AIME and Fuels Division, ASME, meeting. It will be held this year at the Hotel Statler, Cleveland, Oct. 23 to 25. The Industrial Minerals Division will also be having a meeting in the fall but the date and place are not yet known.

In the meantime, coal operators will be interested in the Coal Division-Central Appalachian Section meeting to be held on June 16 and 17 at the Daniel Boone Hotel, Charleston, W. Va. Stream pollution will be a featured subject.

Conveyor Belt Maintenance

by J. R. Thompson

It is common practice, and certainly good business as all of us know, to take care of plant operating equipment. Machinery of any type requires periodic inspection and planned maintenance. With this thought in mind a plant operator should at the end of every shutdown period make regular belt inspections and plan for later appropriate care and maintenance.

The points mentioned below are understood well by all of us but are sometimes forgotten because belt conveyors are comparatively easy to operate. Maintenance of any conveyor belt is dependent upon the fundamental characteristics of the two component parts—cover and carcass.

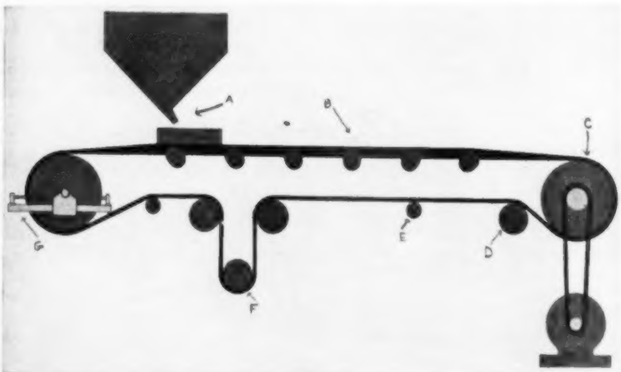
Mr. Thompson is manager, flat belting, industrial products sales, B. F. Goodrich Co., Akron, Ohio.

The interior cotton fabric, cord, steel cable, or synthetic fibre plies supply the main structural strength of the belt. They do the work of supporting and pulling the load, whereas the rubber covers provide resistance to corrosion, abrasion and impact. The load-carrying carcass plies are subject to many hazards such as abrasion, impact and corrosion. The rubber with which they are covered and impregnated is many times more resistant to these damaging conditions. Rubber can stand up well against abrasive action and shock impact as long as the force involved does not distort the rubber beyond its elastic limit. When the rubber cushion cover of a belt is distorted beyond its capacity to yield, the surface is cut or broken and the protective cushion torn away, leaving the interior carcass exposed to damage. Injury and damage are of far greater importance in determining how long a belt will last than the number of tons of material carried.

From the foregoing, then, it can be reasoned that maintenance of a belt actually begins with the belt being specified for the particular job that it has to perform. If a belt has insufficient covers to absorb impact of the job, if the interior carcass will be stressed beyond the limits set up by the manufacturer, the operator stands in serious jeopardy of losing many times the original capital investment in the belt by the smaller tonnage which will be delivered. There are, of course, many other factors contributing to belt life other than cover thicknesses and permissible working stresses. But the point is this: care and maintenance are for getting the long service and lack of same may approach danger lines of operating efficiency and greatly affect possible materials handling savings.

Important machine locations in any given system should now be considered. Principal wear on any conveyor normally occurs at the loading points. It is important to check some of the following points and determine if your conveyor belts are receiving proper maintenance there.

Is the load on a particular conveyor belt dropping from a height greater than necessary onto the belt? (a) Install an inexpensive short feeder belt to take the greatest abuse. (b) Use a pad belt at the loading point for severe impact loading conditions. (c) Several types of shock impact rubber-covered carrying idlers may be used at the loading point. (d) A screen or grid that permits fine material to be placed on the belt first and act as a cushion for larger lumps is standard practice for good loading. (e) For some types of loading the middle horizontal carrying idlers have been removed to eliminate hammer and anvil action.



The points on a conveyor setup mentioned by the author are: a) Loading point, b) Carrying idlers, c) Head pulley and drive pulley d) Snub pulley, e) Return idlers, f) Gravity take-up g) Tail pulley with screw take-up.

Damage, not tonnage, is most important in determining belt life. Check your belts against these common operating errors.

Does your loaded belt sag badly between carrying idlers? (a) This indicates that either the belt does not have proper tension or that the idler spacing is improper. Too much sag causes the load to move with spillage and high horsepower load resulting. Movement of material on carrying idlers may also damage a belt.

Is there proper clearance between the belt and the lip of the chute or the bottom edge of the skirtboards? (a) Make sure that no stationary parts come in scraping contact with the belt surface. (b) Materials to be conveyed should never wedge between the belt or any stationary part. It is suggested that double skirting be used with steel on the inside and rubber skirting on the outside, or giving a labyrinth seal effect. The rubber skirt board strips should flare outward and upward in the direction of material travel in order that the largest size pieces of material will move easily in the direction of flow and not hang under or between the skirtboards. It is important that this rubber strip should be of an all-rubber construction. Old conveyor belting, for example, will pick up sharp particles between the plies and form an abrasive grinding surface if the skirtboard edge should be in contact with the belt.

It is expected that troughing idlers, return idlers, as well as drive terminal, snub or takeup pulleys, be maintained with a clean surface for belt alignment and long life. (a) Terminal Pulleys. Granted that head and tail pulleys are of proper diameter for belt width and thickness the life of a conveyor belt is often endangered by slippage on pulleys and caused by lack of cleaning. Accumulation of such material may cause the belt to run to one side, seriously damaging the belt edge. Material may cling to the belt beyond the drive snub pulley location and then build up on return or takeup pulleys to again cause serious belt misalignment and damage. (b) High tension snub pulleys operating against the dirty side of the belt may be lagged with a ½-in. thick soft rubber lagging with smooth surface. The movement of soft rubber surface tends to brush off material buildup on the belt. (c) A layer of ⅜-in. soft rubber may be vulcanized or lagged to drive pulleys. Lateral or herring-bone grooving of the rubber pulley surface provides not only a surface movement for cleaning, but also a higher coefficient of friction for wet drive conditions. (d) The use of all-rubber wiper blades is usually most effective in conjunction with water spray.

Belt cleaning and pulley cleaning problems go hand in hand and if the operator can keep his belt surface clean after material discharge, pulley cleaning is usually unnecessary. If belts are handling tacky material, some will, of course, pass around the discharge pulley to the return side and continue to cling to the belt. It may carry with it some sharp particles which will be pressed into the belt cover or carcass when the material passes between the belt and high tension snub or drive pulleys. Also an uneven accumulation of the sticky material on the snub, drive, or bend pulleys may rupture the belt carcass. Material which continues to cling to the belt beyond the drive snub pulley will build up on takeup, bend pulleys or on return idlers. This will cause the belt to run off center.

Cleaning devices—For many materials, a water spray along with squeegee wiper blades gives efficient belt cleaning. The water softens and lubricates the material, and rubber wipers can remove it from the belt. The slightly dampened surface then has less tendency for material to adhere to it. (a) **Water spray and wiper**—A large header for several spray nozzles, a hydraulic regulating valve, a control valve for turning the water on and off, a pan to catch and lead off the water from the spray, and the wiper completes the setup. The use of 1.0 in. or thicker all-rubber wiper blades is recommended. The rubber blade should be backed up by a steel or wood bar.

In some cases wipers have been placed at right angles to the belt edges, some have been at a 45°

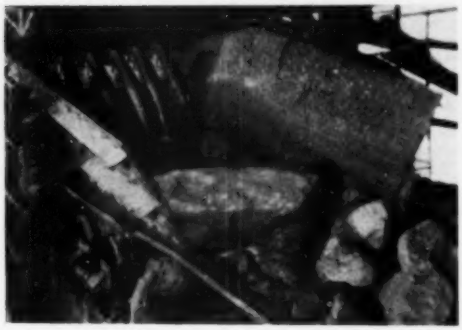
or less angle across the belt and some are mounted on metal frames in groups of two or more shaped like a plow with point in center of belt. All are mounted on springs or pivoted on weighted lever arms to provide pressure between wiper and belt. All cleaning devices are placed as near the discharge point of the load as possible but should not be against the belt where it passes around the pulley. (b) **Water vapor**—for certain types of materials which have a tendency to stick, a fog made by uniting water and compressed air has proved to be beneficial. The water and air mixture blown against the surface of the belt near the loading point and before the material is placed on the belt prevents it from sticking without adding enough moisture to prohibit its use. This cleaner is placed near the tail because its action occurs before load is placed on the belt. (c) **Revolving power driven brushes**—there is usually a high maintenance cost due to expense of bristle or brush replacement. When belt is stationary, the brush tends to take a permanent set and may develop flat spots. Brush cleaners have a tendency to clog and become ineffective. One operator mounted power driven brushes in a group of two. A mounting was made to support combs made from $\frac{3}{8}$-in. thick steel teeth similar to blade guard on a mowing machine. The ends of the teeth touch the bristles of the brushes causing them to flick off most of the dirt sticking to the bristles, thus keeping the brushes clean. (d) **Power driven rubber spiral rolls**—the openings on such rolls tend to become clogged with material. When clogged the revolving roll becomes more of an abrasive wheel. (e) Sometimes **steel blades** are mounted on a flexible arm and placed on a bias angle to the belt travel. These blades often chatter and do not do a good cleaning job. They may even mark the belt cover. Steel bars or blades may be fixed close to belt or mounted on a pivoted weighted arm. Wear is usually quite rapid so that frequent adjustment or replacement is necessary. With a heavy fixed blade there is danger of rock or ore catching and then tearing or gouging the belt. (f) **All-rubber blades** are easiest on the belt but are usually most effective when used with a water spray. The use of old belting is not recommended because the fine material can lodge in the fabric portion and the scraper becomes an abrasive or scouring agent. (g) **Compressed air blast**—several nozzles are usually directed on the belt. It has been successful on a few materials such as wet sand and wet fine anthracite coal.

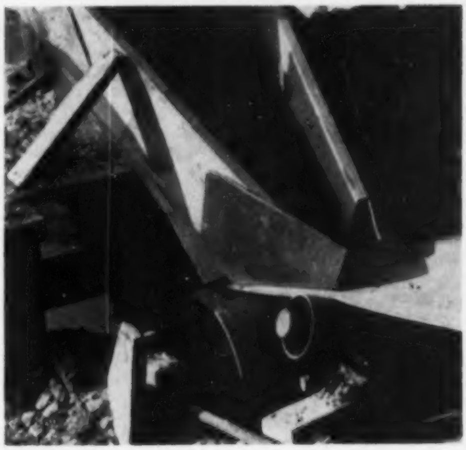
Considerable attention should be given to return idler cleaning inasmuch as these idlers all contact the belt carrying side. As stated before, buildup on return idlers may cause the belt to run off and damage belt edge. Several means have been used to clean return idlers. (a) Carrying idlers. (1) In some conveyor installations there is a tendency for moisture to condense on idlers and combined with ore forms a coating on the moving parts of the idlers. Corrosion of metal parts may result, with an eventual frozen idler to cause misalignment. We have found one plant operator spraying a 0.005-in. zinc coating on the metal rolls to prevent this corrosive action. This thin zinc coating might only have value on the return idlers where abrasion is comparatively low. (b) Return idlers. (1) Rubber covered sleeves have been made to slip over return idlers. If a slight crown is molded into this sleeve, some guiding effect will result along with a cleaning effect of the rubber surface. (2) Return idlers composed of separate pulleys do not permit buildup as easily as a solid surface return idler, but sharp edges of individual pulleys may damage belt top cover if misalignment occurs. (3) A recently developed return idler has indicated good test results. A rubber-covered T cross-section is spirally wound around the return idler shaft to form the pulley surface that supports the belt. The spiral is designed to be left hand and right hand from the center line. The cleaning action has proved quite effective on many jobs.

Crooked Running Belts—A large percentage of premature belt failures is traceable to this condition. Some of the chief causes and remedies for getting belts to run straight are: (1) Improper splicing with mechanical fasteners. Since belt edges are not always perfectly true, it will pay to make sure that the belt ends are square at the splice. This is best accomplished by drawing a long center line 10 or 15 ft back from the belt end. The belt should then be cut at right angles with this center line and the center lines of the two ends matched to make a perfect splice. (2) Aligning frames and idlers. If a belt keeps climbing sidewise on the same idlers, misalignment of several things is possible. First, frames should always be aligned with a transit. It is also good practice to run levels on the opposite side of carrying idlers so that elevations are the same. If it becomes necessary to line up the idlers themselves, work in the direction of belt travel. Usually carrier idlers which are causing trouble are the second or third behind the point where the belt climbs out of line. In adjusting return idlers, it is best to start at the head end of the belt. In all adjustments of idlers, it should be remembered that the effect from idler-shifting is not immediate. Wait several minutes after each change before making any other changes. The belt will shift toward the side where it first touches the idler roll. (3) Tilting idlers. Sometimes a slight forward tilt of the idler carrier helps to keep a belt running straight. The outer axle of the idler, however, should never be moved forward more than $\frac{1}{8}$ in. to $\frac{3}{16}$ in. It should be remembered that *idlers should not be tilted on reversing conveyors.* (4) Frozen idlers. Careful inspection should be maintained at all times to insure against this. Besides causing excessive belt wear, there is a possibility also of actually setting the belt on fire. (5) Insufficient belt contact. Horizontal middle rolls of the idler must make contact with the belt in order to steer it straight. For this reason, belts which will trough perfectly when empty are necessary.

Miscellaneous Maintenance Suggestions—(1) Guard against cover breaks and tears. Many operators guard against intrusion of moisture into the belt carcass by cleaning breaks with a wire brush and brushing on some waterproof belt cement. They have also found it advantageous to paint this onto raw belt ends when metal fasteners are used. Also, such a cement containing a pigment accurately marks places where future,

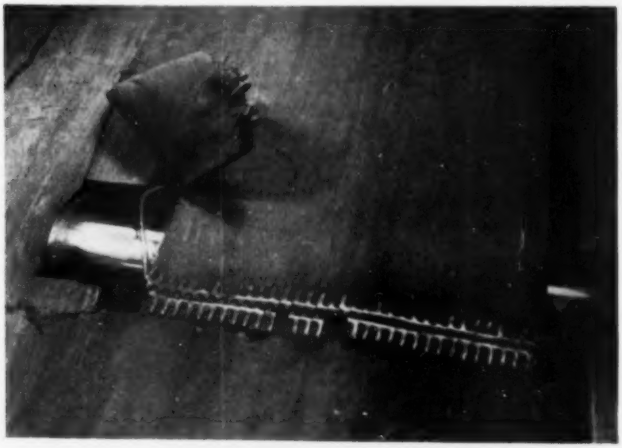
You Can Make Them or Break Them





Above, standard practice for good loading is use of a grid which allows fine material to be placed on belt first, acting as cushion for larger lumps.

Right, the hammer and anvil effect, with loading impact acting directly on pulley and idler, injuring both them and belt.



Left, a ripped belt, the result of improper splicing with mechanical fasteners. Belt ends should be square at the splicing point to prevent uneven pulling at that point.

• • •

Below, left, a piece of old belting has been used as skirting. Sharp particles can accumulate between the plies, causing damage to the running belt. Also, material is running beneath skirting, adding to grinding effect. **At right,** proper skirt boards flare upward and outward, have all-rubber edges which are backed up with steel.

permanent repairs should be made. (2) Make permanent repairs as quickly as possible by means of vulcanized patches. Operators can buy, for a small amount of money, lightweight vulcanizers and repair materials so that permanent repairs can be made when time permits. (3) **Oil on Belts**—Unless the conveyor belt in use has been specified as an oil-resisting belt, the present combinations of crude rubber and American rubber are not oil resistant. Any free grease or oil which gets onto the belt during lubrication of bearings of the idlers or drives, should be immediately cleaned off. Such free oil or grease will cause the rubber to swell and flake off. (4) **Break in New Belts**—After a belt has time to take a "Set," it conforms more snugly to the carriers and runs straight. One good method has been noted for breaking in belts: First, load the belt for one-third of its length and let it stand; then, load for another portion length; and finally, for its remaining length. If this method is used, it is important to have the load properly distributed and the belt in perfect alignment with the carriers. (5) **Light**—Rubber tends to crack after long exposure to direct sunlight owing to the effect of ultra-violet rays, which promote oxidation and hardening of the rubber surface. Modern compounds are made to retard this action, but nothing has been discovered to overcome it. No rubber can therefore be expected to last as long in the open as when protected from the sun. This is a problem in dry climates or high altitudes inasmuch as atmospheric moisture filters out a large part of the ultra-violet rays. (6) **Heat**—Rubber belts stand temperatures up to 150° F and special hot material belts are made to withstand higher temperatures. Any temperature in excess of 100°, however, hastens the oxidation of the rubber. For this reason, some shortening of life is to be expected of belts operating in closed housings in warm weather without ventilation. (7) **Cold**—Low temperatures are practically harmless to rubber, but the accumulation of ice can result in serious damage if it builds up in lumps on the belt surface, particularly on the pulley side. These can be as disastrous as lumps of rock. Ice on the belt surface can cause material to slide on an incline. Wet belts have been known to freeze to pulleys so firmly that the rubber was torn loose from the fabric when the conveyor started. (8) **Moisture**—Even when the rubber covers are intact, excessive moisture in time penetrates to the cotton, causing it to shrink, thereby shortening the belt. The operator should ease off the takeup tension to compensate for this shrinkage. Trouble also results when moisture causes material to cake on pulleys and belt. Wet conditions also cause slipping due to the low coefficient of friction between pulley and wet rubber. Acid water, penetrating injuries in the belt surface, often seriously corrodes cotton before the damage is discovered. (9) **Mildew**—When belts are to be used under moist conditions, the order should specify "anti-mildew treatment". No chemical has yet been discovered, however, to prevent this growth. The safest plan is to keep covers intact by inspection and repair. Mildew does no harm to rubber. (10) **Housing**—The greater ease of inspection and greasing in bad weather makes a housing worthwhile aside from its value as a protection to machinery and belt.

Conveyor belts are best repaired by electrically-heated vulcanizers of the appropriate size to accommodate certain width belts. Vulcanized repairs can be made by any good plant maintenance man after a few hours of instruction. Repairs usually require replacement of broken carcass plies and/or missing cover patches with new unvulcanized stock. Belt splices are made in much the same manner as repairs using a conventional step-splice design. Literature for both these operations is available listing vulcanizers, tools, repair materials such as friction fabric, cover stock, cements, etc. A well-balanced and economical plant repair program is the best insurance that any plant can use to reduce its materials handling cost per ton.

Belt repairs can be made by plant maintenance men. Relatively inexpensive vulcanization devices are available, and they ensure good repair work. Much literature is available on the subject.

Ferroalloys in 1949

by R. G. Knickerbocker

A MOST imoprtant research and development item on ferroalloys in the calendar year of 1949 was the increase of interest in the recovery of secondary manganese. Owing to the importance of manganese to the nation's welfare, it is most necessary that all possible avenues of research and development on the recovery of this vital metal be investigated.

The principal objective of this new development is to recover manganese from steel furnace slags. Steel furnace slags contain a total of approximately 772,000 tons of metallic manganese which is compared to the 680,000 tons of metallic manganese as ferromanganese that is estimated to be used by the steel industry, annually, at this time.

R. G. Knickerbocker is Metallurgical Engineer at the Bureau of Mines, Rolla, Mo. This paper appeared originally in the Journal of Metals, March, 1950.

To those who are interested in the recovery of primary manganese from low-grade domestic ores containing less than 35 pct manganese, it will be encouraging to know that these slags, containing 5 to 9 pct manganese, are being subjected to considerable research and development work.

The low heat of formation, low viscosity and low specific gravity characteristics of the manganese sulphide-oxide mattes were utilized in Europe during World War II in the treatment of slags and low-grade manganese materials for the recovery of manganese. This is one avenue of investigation of possible value in the program of recovery of domestic primary and secondary manganese.

Results of research work on the influence of temperatures above 1350° C indicate that the affinity of sulphur for copper, manganese and iron is in the order given. Attention is called to the fact that the sulphur in the steel bath is higher when the scrap charge contains copper because of the removal of sulphur from the furnace atmosphere by the copper. That brings up the question of the solubility of copper and manganese sulphides.

The Federal Bureau of Mines initiated new work on the conservation of manganese during the year. Three targets were set up, (1) reduction of sulphur in steel to reduce the necessity for manganese, (2) recovery of manganese from openhearth slags, and (3) development of substitutes for manganese in the making of steel. The steel producers through the American Iron and Steel Institute have furnished the Bureau with $50,000 for these investigations.

A very valuable contribution by the Carnegie Illinois Steel Corp. was made this year covering the details of research work on the removal of sulphur, the effect of the scrap charge, the use of flush-off slags, the importance of distinguishing between the sulphur as introduced by the scrap charge and that introduced in the pig-iron charge in the openhearth; the effect of time between the pig-iron tap and the openhearth molten bath as regards the sulphur content and the use of manganese ores in the blast furnace. The statement was made that the mechanism for the removal of sulphur was as manganese sulphide in the flush-off slag.

Work on a process for recovering manganese from low-grade domestic ores by leaching with waste pickle liquor, which is produced in large volume in certain steel operations, was reported.

The Standard Mining Co., a subsidiary of Standard Ore and Alloys Corp., started to process manganese ore in the Cushman field near Batesville, Ark., during 1949. The "Potashnick" gravity concentrator which operated during World War II is being utilized for the milling of low-grade manganese ore.

Production of ferromanganese at the Anaconda Copper Mining Co.'s new ferroalloy plant in Anaconda, Mont., was initiated in February 1949.

Electrolytic manganese production and sales were not so good as in 1948 but better than any year prior to 1948. Although no new uses were developed for electrolytic manganese during the past year, the increasing acceptance of the higher purity metal in established fields, particularly for stainless steels, is noteworthy at this time.

The seriousness of the manganese supply problem was recognized by the highest levels in government and in March 1949 the National Security Resources Board, at the request of the President, established the Interdepartmental Manganese Coordination Committee with James Boyd, Director of the Bureau of Mines, as chairman. This committee was assigned the responsibility of coordinating efforts of industry and government to attain a satisfactory balance between total domestic requirements and supplies from all sources.

The imports of manganese to the United States in the first ten months of 1949 from Russia were only 6 pct of the total manganese imported into the country.

In terms of manganese ore plus the ore equivalent of ferromanganese, the African Gold Coast was the largest source in 1949, furnishing approximately 450,000 short tons. India was expected to supply approximately 425,000 short tons and the Union of South Africa is estimated to exceed 300,000 short tons. Inadequate transportation facilities held Brazil's shipments to 144,000 short tons. Total imports of manganese ore during the first ten months were 1,201,240 short tons. Domestic production of manganese ore is estimated as 115,000 short tons—which is slightly less than 1948. Shipments of ferruginous manganese ore and manganiferous iron or containing 5 to 35 pct manganese are estimated at 1,000,000 short tons during 1949.

Ferromanganese consumption decreased 15 pct in 1949 approximating the extent of the lower steel production rate. Imports of 60,000 tons of ferromanganese were received this year.

The price for standard manganese ore rose from approximately 70¢ per long ton unit to about 82¢ in November; standard ferromanganese rose from $160 per gross ton, in carload lots, to $172 per ton in November.

Ferrochromium

A new chemical method for the treatment of low-grade chrome ore has been developed in Australia which makes use of a sulphuric-chromic acid solvent and electrolytic oxidation. The product contains 95 pct Cr_2O_3 and represents a possible recovery of 75 pct of the chromium in the ore.

A surface-chromizing process has been introduced into this country by the Diffusion Alloys Corp. of New York. This is a British metallurgical development. A pack-carburizing method uses containers with a ferrochromium base material and a suitable catalytic agent. Temperatures range from 1700° to 1850° F, depending upon the carbon content of the steel to be carburized. Chromium penetration in low-carbon steels is about 0.03 to 0.35 in., while in high-carbon steels penetration is 0.003 to 0.005 in.

Important data pertaining to chromium-carbon oxidation and temperature relations have been reported this year by the Union Carbide & Carbon Research Laboratory, Niagara Falls.

The Mead Ferrochrome reduction works of the Chromium Mining & Smelting Co., in the state of Washington, offered to purchase domestic chrome ores during the year. Minimum specifications were 45 pct Cr_2O_3 content, 2.5 to 1 chrome-iron ratio. This does not allow the use of the submarginal Montana chrome ores since the chrome-iron ratio in these concentrates is not high enough. Considerable investigation of the beneficiation of Montana chromite by reducing roast and sulphuric acid leaching of the iron to correct the ratio discrepancy has been conducted.

Some of the African Transvaal chrome ores are reported to require beneficiation of a similar nature and although the detail of the process has not been given, the cost has been stated as approximately 15 shillings per ton.

Imports of chromite during the year were nearly twice the consumption rate and are estimated to have totaled 1,200,000 short net tons. Philippine Republic supplied 30 pct of the total chromite imports and was the largest exporter of chromite to the U. S. Nearly all of the Philippine chromite was of refractory grade. Turkey supplied 24 pct of the total, all of which was metallurgical grade. Domestic production of chromite was estimated to be 300 tons.

The ferrochromium consumption for the first 9 months of 1949 was at a rate of 93,800 tons per year compared with 122,753 tons in 1948.

Cobalt

The Rhokana Corp. in Northern Rhodesia expects to double its present output of 362 tons of cobalt per year. They plan to produce electrolytic cobalt.

The Howe Sound Co. of New York continued exploration and development work on their cobalt property in Idaho and there has been an appreciable increase in the ore reserves there as a result of this work. Preliminary construction was initiated and is planned to be continued through 1950.

The International Nickel Co. of Canada is reported producing at the rate of 15 tons of cobalt monthly. This is cobalt from the nickel ore of the Sudbury Basin.

The Falconbridge Nickel Mines, Ltd. report that they will be producing substantial quantities of cobalt in 1951. This is also from the Sudbury Basin nickel ore.

The Reduction & Refining Co., Kenilworth, N. J., is processing government-owned cobalt ore and subgrade metal.

Union Miniere du Haut Katanga, Belgian Congo, is enlarging their concentrators and the electrolytic plant for copper and cobalt. Hydroelectric power developments are being enlarged to take care of this work.

The Bethlehem Steel Co. was again the only producer of commercial domestic cobalt ore in the United States.

Imports of cobalt into the U. S. for the first nine months of 1949 were 15 pct less than in the corresponding period of 1948.

Effective April 1, 1949 the price of cobalt metal —97 to 99 pct cobalt—was raised to $1.80 per lb.

Exothermic Alloys

Considerable progress has been made in the use of exothermic alloys; they have a definite place in the steel industry and utilize low-grade domestic ores. The use of high-carbon "Chrom-X" now amounts to approximately one-third of

the high-carbon ferrochrome used. A "Chrom-Sil-X" is finding considerable use in the cleaning of slags and the making of stainless steel.

Nickel

A report was issued this year on the five years' operation of the Nicaro Nickel Project, a wartime project in Eastern Cuba initiated in February 1942 and concluded in March 1947. Production amounted to 63.5 million lb of nickel chiefly in the oxide form. This oxide was acceptable for use in the iron-steel industry and as such was a substitute for ferronickel.

The International Nickel Co. of Canada reported in June this year a geological exploration revealed nickel was widespread in the Lynn Lake area of Manitoba and two provinces of Venezuela. A new ore body at the Creighton mine, Sudbury district, was largely responsible for increasing ore reserves at the end of 1948 to the all-time record of 246,177,000 short tons.

Imports from Canada, the chief source of supply, were 8 pct less in the first 10 months of 1949 as compared to 1948. This was because of the substantial decline in the output of nickel alloy steels, the steel strike and the production loss in stainless steel.

Molybdenum

Numerous research and development programs have been proposed and are active both on molybdenum metal and as an alloying element to conserve nickel, chromium and manganese in steels. The later programs are based on the plentiful supply of molybdenum which is produced from extensive reserves in Colorado and as a by-product from the copper industry, coupled with the highly satisfactory service the molybdenum alloy steels gave during the recent emergency. Most of the NE or national emergency steels contained 0.15 to 0.25 pct molybdenum.

Advancements made in the melting and fabrication of molybdenum metal has progressed to a commercial stage and the metal is being processed into chemical ware as a replacement for platinum crucibles, etc. Considerable interest continues in the use of molybdenum and high-molybdenum alloys for high-temperature applications since the strength (over 100,000 psi) of the metal does not materially decrease at temperatures as high as 1600°F.

Consumption of molybdenum was 16,201,800 lb in the first nine months—a drop of 8 pct as compared to the same period in 1948.

Tungsten

The Bradley Mining Co. has completed its new tungsten mill to serve the Ima Mine in Idaho. The plant is expected to be in full production during 1949.

Important deposits of tungsten and molybdenum in Guanajuato, Mexico have been discovered.

Brazil has reported possibilities of increasing its production of tungsten, beryllium and tantalite in the northeast section.

There was a 28 pct loss in consumption of tungsten concentrates in the first nine months of 1949 as compared to the same period of 1948. The Tungsten Mining Corp. of North Carolina was the largest domestic producer.

Columbium, Tantalum and Vanadium

The Electro Metallurgical Div. of Union Carbide & Carbon Corp. reports new and increased uses for ferrocolumbium have made it necessary to determine the extent to which tantalum, vanadium and other carbide-formers could be used together with, or in place of, columbium. It has been found that tantalum can be satisfactorily used in the high-temperature alloys where strength at temperature is the major consideration. Vanadium can also be used, but the oxidation resistance of the alloy is affected adversely if the vanadium addition exceeds about half of one percent. The use of tantalum and vanadium for purposes of eliminating susceptibility to intergranular corrosion is possible but these elements are not as effective as columbium. Information on these subjects was available many years ago but had to be reconsidered in the light of new developments and new alloy compositions now being used. Stainless steels very low in carbon (0.03 pct maximum) have proved to be suitable for many uses where the grades containing columbium or titanium were formerly used.

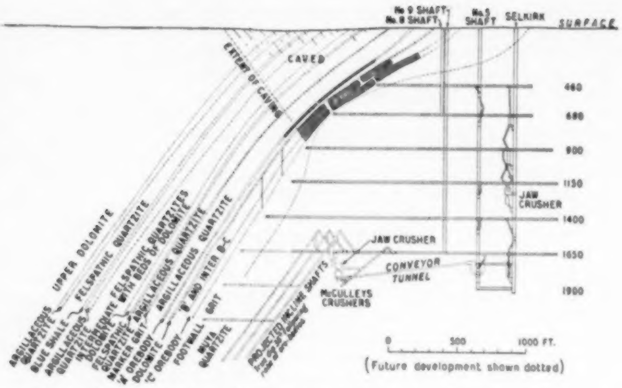
Titanium

The forecast potentialities of titanium and titanium alloys in the not too distant future are of particular interest to the ferroalloy industry. The principal characteristics of the metal are its attractive weight: strength ratio and outstanding corrosion resistance. In the latter respect it compares favorably with platinum as shown in various sea water tests. This new commodity may be expected to gradually replace some of the stainless steels in special applications. Added interest in titanium relations to the steel industry arises from the development of the 36 pct iron ores (containing 32 pct TiO_2) of the Allard Lake area. This reserve of iron ore with titanium byproducts is being processed by the Quebec Iron & Titanium Corp. Its production plant is reported to be able to turn out 175,000 tons of high-grade pig iron and 225,000 tons of titanium slag annually, and tentative plans have already been made to expand capacity.

At least until the titanium commodities have become well established, the metal will remain closely allied with paint pigments, electrode coatings and ferroalloys. In this connection the National Lead Co. acquired certain assets of the Titanium Alloy Mfg. Co. during 1949. Also during the year, Allegheny Ludlum Steel Corp., E. I. du Pont Corp., and Dominion Magnesium, Ltd., announced the production of titanium-metal ingots weighing 25 to 400 lb.

Estimated domestic production of rutile in 1949 was 11,600 tons. Imports of rutile from Australia were 2750 tons for the first 10 months of the year, or slightly more than one-third of receipts for the same period in 1948. Ilmenite prices decreased from \$20 per ton to \$14-16 during the year. Further exploitation of titanium metal may be expected to materially benefit the economic and production status of these minerals.

Mining Methods at *MUFULIRA*



Section through shafts shows three ore bodies being mined and hanging wall beds.

by J. P. Norrie and W. T. Pettijohn

THE Mufulira Copper Mine, Ltd., is in Northern Rhodesia, ten miles from the Belgian Congo border, and is one of the group of four operating mines comprising the Northern Rhodesian Copperbelt.

The ore deposits at Mufulira are on the limb of a large syncline striking NW.-SE. and dipping NE. at about 45 deg towards the Congo border. The host rocks are well-consolidated, highly-silicified, felspathic quarzites of the lower part of the pre-Cambrian *Séries des Mines* of the Katanga. The ore rocks are underlain, unconformably, by the older "Muva Series," in this case hard, green, metamorphosed quarzites.

There are three superimposed orebodies, the first, or upper, orebody averaging 40-ft true thickness with a strike length of about 5000 ft, the second, or middle, orebody of about 50-ft true thickness with a strike length of about 6000 ft, and the third or lower orebody of around 60-ft true thickness with a strike length of 8000 ft. The orebodies are separated by varying widths of waste, or very low-grade ore, from a few feet up to 50 ft. The middle and lower orebodies are so close together for a strike length of 3000 ft that they are mined as one orebody. In all, the mineralized zone is about 200 ft in true thickness.

Mineral distribution is remarkably even, both across the width of the orebodies and along the strike. There is, however, a falling-off in width and amount of copper mineralization towards the fringes, particularly to the west; so much so that when the mineralized width has dropped to about 10 ft the copper content is not commercial.

Mineralization, occurring as bornite, chalcopyrite, and chalcocite (in order of importance), consists of disseminated particles with occasional stringers and veinlets cutting across the bedding or in joints and other lines of weakness. Malachite, cuprite, azurite, chrysocolla, and native copper are found, but not in quantity. Down to about 900 ft below surface there is considerable secondary enrichment, generally chalcocite.

This article is condensed from a paper in the December Bulletin of the Institution of Mining and Metallurgy. Mr. Norrie is a consulting mining engineer and Mr. Pettijohn was formerly mine superintendent for Mufulira Copper Mines, Ltd., and is an AIME member.

The orebodies, as such, do not outcrop. Commercial ore ceases at from 150 ft to 350 ft vertically below surface. There is a pronounced rake to the north. Diamond drilling has outlined the ore down to 3000 ft depth and the proved ore reserves now stand at 86,233,000 tons of 4.05 pct total copper and 0.07 pct oxide copper; the actual reserves are probably much greater.

The immediate hanging-wall of the mineralized zone is a 200-ft thick band of argillaceous quartzite overlain by dolomite shales and quartzites, all heavily water-bearing. The immediate footwall of the lower orebody is a cross-bedded sandstone, or grit, often poorly consolidated, sometimes water-bearing, but generally improving in depth. It is in this grit that haulages, orepasses, and other development for ore-gathering must be excavated. The underlying Muva quartzite, already mentioned, is in such a position as

to be the rock generally selected for shaft sinking, pump stations, crusher stations and other big excavations, and for this purpose it is ideal.

The current mining methods are: (1) semi-shrink or back stoping; (2) sub-level stoping (conventional); (3) sub-level stoping (continuous retreat), and (4) block-caving.

Semi-Shrink or Back Stoping

This method is used in the single orebodies, where the ore thickness is less then 25 ft. Ladderway raises are run from one haulage level to the next above at 455-ft intervals along the strike, with mining raises between at 75-ft intervals, giving the stope a dimension of 60 ft on the strike by 300 ft or over on the dip by the ore thickness of 25 ft or less (Fig. 1). Where ground conditions require it the strike width of the stope may be reduced to 40 or 30 ft and the dip length shortened. Pillars of 12 to 15 ft are left between stopes.

Ore chutes on the haulage are placed at 75-ft intervals and connect to a grizzly drive 40 ft above, where crosscuts are run into a grizzly or bulldozing chamber below each stope. A double-ended three-bar grizzly, with a 24-in opening, is fed by ore drawn from two cones serving the 60-ft stope. The next level up the dip is the cone drive, about 25 ft higher vertically than the grizzly level, and this is used for opening up the stope face. A parallel drive nearer the hangingwall is used for scraping when dips are flat and the ore will not run. Other sublevels, run for convenience in mining and for ventilation purposes, connect the mining raises to the ladderways at intervals of from 60 to 90 ft on the dip.

When development is well advanced, and after ladderways have been equipped and grizzlies installed, stoping operations commence with the cutting of the cones between the grizzly level and the cone drive. This is done by stoper machines shrinking up on the ground and retreating from the cone raises. As soon as the cones are complete, a square face across the complete 60-ft strike width of the stope is established. This is carried up dip using stopers to drill off the broken ground, which has to be drawn daily to take care of the swell and mucked back from the face because of the flat dip. As most hanging-wall rocks are strong and ore thicknesses are not over 25 ft, little difficulty is encountered in maintaining a safe mining face.

Access to the stope, while mining, is obtained through the various sublevels from the main ladderway raise, and a ladder is also maintained in the mining raise from the stope face to the sublevel above.

This class of stope normally requires one European miner, who supervises a gang of about seventeen Africans—five machine crews, a boss boy, and a helper. Production varies with stope widths and conditions, but 6000 tons per month is a fair average.

Good ventilation is obtained by bringing fresh air from the main ladderway, through the sublevel in use, to the end of the stope where it can sweep the whole face, blowing air into the mining raise which connects directly to a return airway at the top of the stope.

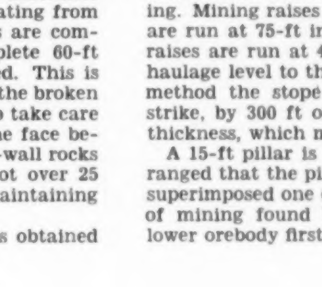
Sublevel Stoping (Conventional)

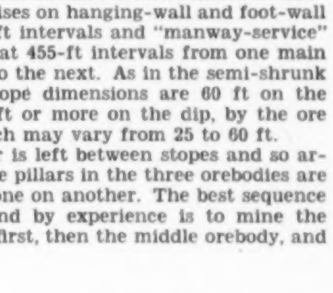
This method was first introduced to this district at the Roan Antelope mine in 1931 and is a method that was used in all the mines of the Copperbelt and is, therefore, described here as conventional.

The main haulage level, as shown in Fig. 2, is run in the footwall about 40 ft from the orebody. Then, 40 ft above the main haulage, two drives are run, one in ore for entrance to the grizzly crosscuts and one in footwall waste for exhausting blasting fumes. Grizzly crosscuts are situated at 75-ft intervals along the strike, corresponding to the chute raises from the main haulage. Next, 25 ft above the grizzly drives, there are two sublevels, one run over the centers of the cones, or drawpoints, and one run farther back in the hangingwall. From this second drive crosscuts are run through to the cone drives at the center of the drawpoints and scrapers are set up here when the dip is too flat for the ore to run down the footwall of the stope. Above the cone level sublevel drives are spaced in the center of the ore at 28-ft vertical intervals for convenience in mining. Mining raises on hanging-wall and foot-wall are run at 75-ft intervals and "manway-service" raises are run at 455-ft intervals from one main haulage level to the next. As in the semi-shrunk method the stope dimensions are 60 ft on the strike, by 300 ft or more on the dip, by the ore thickness, which may vary from 25 to 60 ft.

A 15-ft pillar is left between stopes and so arranged that the pillars in the three orebodies are superimposed one on another. The best sequence of mining found by experience is to mine the lower orebody first, then the middle orebody, and

Fig. 1. Semi-shrink or back stoping is used where ore thickness is less than 25 ft.





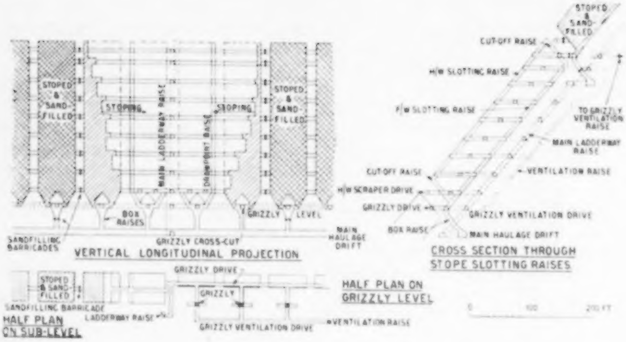


Fig. 2. Sublevel stoping followed by sand filling was first used in 1931 and was widely adopted in the Copperbelt.

then the upper orebody, with sandfilling introduced following the same sequence.

The stope is opened by belling out the cone from the grizzly level to the cone drive, using stoper machines. As soon as the first cone is completed, a slot is cut at the end of the stope, removing the rock between the hangingwall and footwall mining raises. This is usually done by stoper machines, shrinking upwards on the broken ground from the cone level right to the top sublevel of the stope. This operation is a slow one and the tonnage gained from shrink-swell is about 2000 tons per month. After shrinking is completed, trails are run to the hangingwall and footwall on the various sublevels and benches prepared for blasting. The trailing work is done by 3½-in. drifter machines, the drilling of down holes on the benches by 3-in. jackhammers, and the up holes by 3⅛-in. stoper machines. The average depth of hole drilled in stopes is about 8 ft. This operation of benching and trailing continues towards the ladderway until the stope face reaches the 15-ft pillar, which is left between all sublevel stopes.

Each stope is run by a European miner, supervising a crew of thirteen Africans running five rock drills. The average monthly tonnage obtained from this type of stope is about 6000, including the slotting period.

Ventilation is maintained, as in the semi-shrink method, by introducing fresh air, obtained from the ladderway intake, by means of a 14-in. ventilation pipe, with a fan blowing on to the stope bench. The mining raises are connected to a return airway in the crown pillar at the top of the stope, where the vitiated air carries on through the return air raises to the main exhaust fans and is not used again. As already mentioned, a grizzly ventilation drive, run below the footwall of the stope, is used to exhaust secondary blasting fumes and this opening also connects with return air raises and the exhaust fans and is the return airway for stopes coming from the next lower haulage level.

Sandfilling is still used in conjunction with sublevel stoping and, in the lower orebody, not more than two stopes are allowed to stand open without fill. In the upper and middle orebodies not more than four stopes are allowed to stand open without fill. Unless this practice is followed fairly closely stopes cave without much warning, causing air blasts, and also tearing into adjacent unmined stopes, causing losses in extraction. The cost per ton of sand placed as fill is at present about $.08.4.

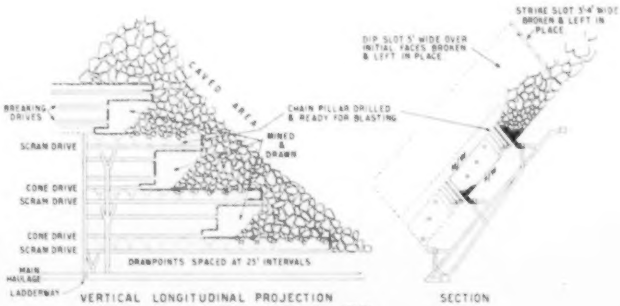
Sublevel stoping costs per ton, excluding development, vary from about $.86 to $1.18, according to the amount of tonnage mined, the width of ore, and other local conditions.

Extraction from this class of mining varies from 60 to 80 pct depending on ground conditions.

Sublevel Stoping (Continuous Retreat)

This system of mining is used in the upper orebody, in the sections over the "block-caving"

Fig. 3. Sublevel stoping is also used over caving areas without sand filling where it is desirable to have the caving initiated.



areas, where it is desirable, in any case, to have the caving initiated and where sandfilling would be more of a hindrance than a help.

In this method there is a series of three open sublevel stopes with back lengths of about 85 ft between main haulage levels. These stopes retreat continuously along the strike, leaving crown pillars of about 30 ft, which are blasted out in sections of approximately 70 ft along the strike.

To start the caving of the strong hangingwall rocks in this mining area a dip slot of about 300 ft back length by 3 or 4 ft wide, extending about 90 ft into the hangingwall, and a strike slot of similar dimensions, at right angles to the dip, and also in the hangingwall, are cut. It is thought that the strike length of this latter slot should be several hundred feet, but this point has not yet been settled.

Draw from the stopes is through footwall openings to scraper drifts located below each crown pillar, where the ore is scraped to ore-passes spaced at 200-ft intervals.

The rock-breaking operation carried on in the individual sublevel open stopes is exactly the same as that for conventional sublevel stoping.

Pillar drilling is done by high-speed diamond drills, core-drilling $1\frac{7}{16}$-in. diameter holes varying in depth from 17 to 50 ft. These holes are drilled from crosscuts, in patterns as near as possible parallel to the strike, in order to blast the rock down to the drawpoints before caving waste rock runs in, in the interests of better extraction.

The anticipated extraction from this method of mining is about 85 pct and it is expected that it will supplant sublevel stoping (conventional) methods in the other separate orebodies after more experience has been gained. Mining costs per ton for this class of work are expected to be less than those obtained in other sublevel methods because the slow and costly slotting process is partly eliminated.

Ventilation arrangements for continuous retreat are similar to those for the conventional sublevel stoping.

Block-Caving

This method of mining is applied to the middle and lower orebodies where they run together over a strike length of about 3000 ft and the true thickness varies from 100 to 120 ft. Two retreat faces are in the process of being established, one in the west, between the 660-ft level and the 1400-ft level, and one in the east, between the 900-ft level and the 1400-ft level. Blocks are approximately 200 by 200 ft in plan, and the sequence of mining is on an offset checker board pattern.

An ore-pass and ladderway system is run in the footwall waste rock from one main haulage level to the next at 200-ft intervals along the strike, in the center of each block. Scraper levels are established at 25-ft intervals vertically. About 15 to 20 ft under the orebody footwall, scram drives are run along the strike. Branch ore-passes connect each scram drive to the main ore-pass, which transfers ore to the main haulage level below. On either side of the ore-pass, in each scram drive, four drawpoint raises are cut at 25-ft intervals into the footwall of the ore. This development scheme thus forms a 25 x 25 ft horizontal pattern of drawpoints, with a 100-ft maximum length of scrape to the ore-pass, which is bridged with a heavy steel 24-in. opening grizzly, carrying a 30 or 50-h.p. double-drum scraper hoist.

After waste development is completed the development in ore is carried out, first running the cone or undercutting drives, which follow the footwall contours, connecting up drawpoint raises. Boundary-weakening crosscuts are then run from footwall to hangingwall at the strike ends of the block. Weakening drives are also run along the down dip side or north boundary of the block, 25 ft vertically over one another. More recently this ore work has been accomplished by entering this development area from the upper orebody in advance of any waste work done from the footwall side. This defines the block limits and establishes an accurate footwall before drawpoints are laid out. Once a block is established the development continues along the strike, at a rate determined by the production required from the particular face.

Initial blocks require extra boundary development extending into the waste between the middle and upper orebodies. A dip slot is cut on one side extending from the hangingwall of the middle orebody to the footwall of the upper orebody, and a strike slot is cut at the top end of the block covering the same vertical distance. This work insures that the arch will break as undercutting progresses.

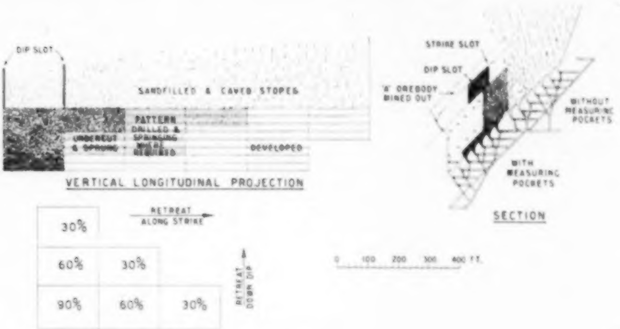


Fig. 4. Block caving is applied to the middle and lower ore bodies where they run together making a thickness of 100 to 120 ft.

Undercutting is started at the top corner of the block, working down dip and retreating along the strike from the cave to the solid in each case. Stoper machines are used carrying inclined shrink stopes of 6 to 8 ft in width from one cone level to the next, forming a checkerboard pattern. As undercutting nears completion at any particular horizon a certain amount of additional breaking is done in the boundary crosscuts and drives, the backs being ripped down for 10 ft and the "solid" corner cut through from level to level.

The results of cantilever action breaking up the block can be observed as the undercut progresses. Bedding planes tear apart and joint planes open up. When natural caving forces come into action the apparently massive quartzites break up with much conchoidal fracturing, the cracks caused by undercutting appearing almost vertically above the working face in the various sublevels as the mining progresses down dip. Heretofore, at this stage, the blasting of sprung holes, in order to induce the cave, was carried out, but recent observations show that no more than undercutting and boundary weakening is generally required.

After the completion of the undercut work, a slow and carefully-controlled draw is started, favoring the down dip or solid side of the block slightly. Drawing at this stage is at the rate of about 3-in. per day and this continues until the block is about 30 pct drawn. The rate of draw is then stepped up to about 4 in., or a maximum of 6 in. per day, until drawn to completion. The differential in draw allowed between adjacent drawpoints is 5 pct, between adjacent scram drives 10 pct, and between adjacent blocks 30 pct. Control of the draw is very important and this work is under the supervision of engineers in the mine technical department, who give the draw orders which are passed on to the operating officials.

Data from draw reports is recorded daily. From these reports profiles of the various sections through the block are kept up to date. In addition to this measuring of the ore from each drawpoint, regular grab sampling is done and geological observations made to see that the ore is being drawn properly. When a block is nearing completion, low assays determine when a drawpoint should cease to be drawn.

Various attempts have been made to remove the human factor from draw control as much as possible. Draw sheets calling for a variety of scoopfuls from different drawpoints were not easy to check. It was found better to call for certain drawpoints to be pulled and record the scoopfuls of ore, there then being less chance of paper-perfect reports. Adjustments to draw are made on a shift-to-shift basis; the number of cars filled acts as an overall check.

Certain blocks were prepared with short raises or measuring pockets between the drawpoint and the scram drive. Each measuring pocket has a round-timber grizzly on top to control the flow of rock. The pockets were filled on one shift, the drawpoint closed off, and the measured rock scraped on the following shift. That these measuring pockets were not the success that had been anticipated is attributed to the very variable rock column above the drawpoint. Some drawpoints drawing, for example, from the inter-orebody rocks would run like water and flood the scram

Fig. 5. Boundary weakening crosscuts are driven from foot to hanging wall at the strike end and along the down dip side.

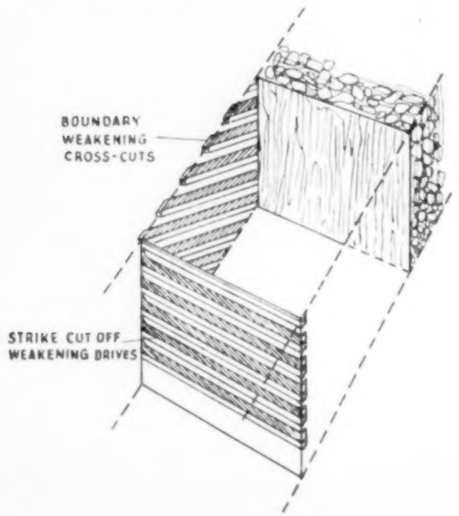
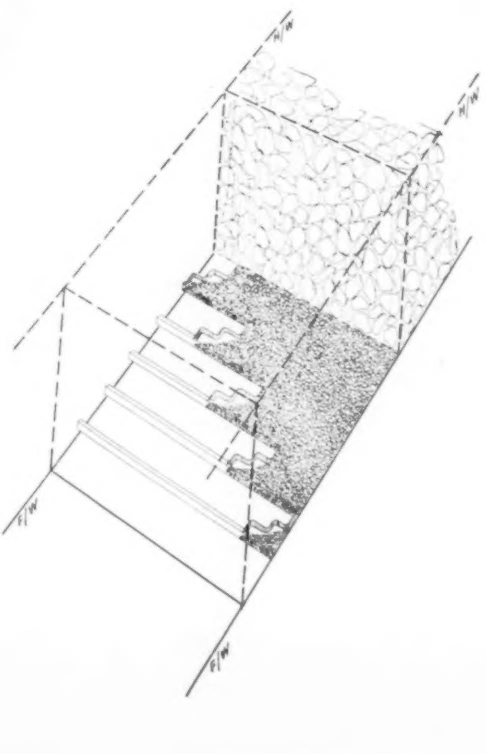


Fig. 6. Undercutting retreats down the dip and from the cave to the solid.



drive, while others in graywacke would fill the measuring pocket with large chunks. Recourse had then to be made to scoop counting. Control devices might be installed at the lower end of the measuring pockets, but with a maximum 7000 tons to be drawn from a drawpoint there is a limit to expenditure. Draw control is a problem still awaiting a satisfactory solution.

A series of experiments on caving and drawing was run, using a glass-sided model to a scale of 1 to 100. The rock was crushed to the approximate scale size and fritted together by means of a gypsum-base cement. The experiments with simulated sandfilled stopes above the draw area were particularly useful. Visual inspection of the progress of these experiments was of much value in instructing the operating officials.

The average monthly production from a block is 20,000 to 25,000 tons and this is obtained on two scraping shifts. A scraper crew for one block consists of eight Africans, with one European supervising two crews. Secondary blasting powder consumption is at the rate of about 1 lb per 3 tons of ore.

Throughout the development of the block caving method close contact was maintained with the Climax Molybdenum Co., and exchange visits between engineers of the two properties took place. Such items as mapping of caveability and the study of rock-behavior by means of scale tests were copied directly, if with some modification, from Climax. The Climax engineers deserve full credit for their share in developing a mining method that appears to meet the case.

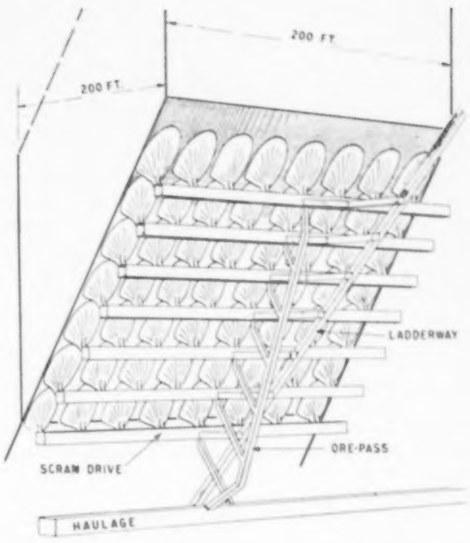


Fig. 7. The gathering system is driven in the footwall waste rock.

Sintering Adirondack Magnetites

by W. R. Webb
and
R. G. Fleck

THE concentrate produced from the Adirondack magnetites is, of course, too fine for direct use in a blast furnace and must be agglomerated before it can be considered a blast furnace ore.

The four operations in the area use sintering —three, including the Jones & Laughlin operation, employing the Dwight Lloyd continuous type machine and one operation using the Greenawalt batch type process. Each operation uses fine anthracite coal, either No. 4 or No. 5 buckwheat size, for the mix fuel.

This paper is a continuation of "Beneficiation of Adirondack Magnetite," which was presented by Messrs. Webb and Fleck in the April issue of Mining Engineering.

A typical sieve analysis of the coal used at Benson shows—

Mesh	%	Cumulative percentage
+14	5.2	5.2
+20	19.2	24.4
+28	32.4	56.8
+35	24.4	81.2
+48	11.6	92.8
+65	4.8	97.6
+100	2.0	99.6
—100	.4	100.0

Four storage silos with a capacity of 500 tons of concentrate or 140 tons of coal serve each sinter machine. The sintering operation consists essentially of mixing concentrate and anthracite coal in the proper proportion and burning out the coal to generate sufficient heat to result in a partial fusion and thus produce the cinder type material called sinter. The amount of fuel required will vary with the size of fuel used and is dependent upon the physical quality of the sinter required.

Each Benson machine consists of 125 pallets measuring 2 x 6 ft which operate on wheels on a 70-lb. rail track to form a continuous conveyor 6-ft wide and with 102 ft 8 in. of effective sintering length. Semicircle closed guides at each end of the machine serve the same purpose as the head and tail pulleys on a conveyor and retain the pallets as they are being inverted for the return cycle. A dc motor drive permits a linear speed range of 65 to 130 ipm. Pallet bottoms are made up of individual finger type grate bars to allow ¼-in. x 20-ft openings which produce a total air space amounting to 14 pct of the grate area.

Fourteen wind boxes provide a continuous suction under the 102-ft-8-in. effective length of the machine. A fan with 107-ft diam impeller and rated at 144,000 cfm at 24-in. water suction provides the draft for a machine.

In addition to the raw concentrate and coal used in the sinter machine feed, small pieces of sinter amounting to 30 pct of the total mix, which have been removed from the sinter product by a fixed 1-in. opening grizzly are returned with the new feed for the double purpose of resintering the extreme fines and to improve the permeability of the bed to increase sintering rate.

The sequence of the sintering operation can be reviewed as follows: The fine anthracite coal, approximately 5 pct of the total mix, is added to a 24-in. belt conveyor at a controlled rate by a vibrating feeder. The concentrate representing 65 pct of the total mix is distributed over the coal by a table feeder. The hot returns are added by vibrating feeder after the concentrate which acts as a cushion to protect the belt from the hot addition.

The feed is transported to a double shaft pug mill for mixing and is discharged from the pug mill directly into a conventional swinging spout which distributes the mix across the 72-in. width of the machine. Normally a small measured amount of water is added to the mix in the pug mill to bring the moisture content of the mix into the range of 6.5 pct to 7 pct which has been found to be the optimum for best bed permeability. The feed rate is synchronized with the machine speed to control the depth of bed, so that the bed can be leveled by a strike plate with a minimum of compacting. Bed depth is normally maintained at 8½ in.

Immediately following the loading point, the pallets travel through an oil-fired ignition furnace of local design. The furnace is 6 ft long and consumes approximately one gal of fuel oil per ton of sinter produced. The purpose of the furnace, aided by the suction from the wind box below, is to uniformly ignite the mix coal in the top of the bed.

As the ignited bed moves out of the furnace toward the discharge end of the machine, the mix coal continues to burn from the top downward, supported by the air which is drawn through the bed for its entire length by the suction system. The air which is drawn through the bed naturally pulls out some fine solids. In order to remove this dust ahead of the suction fan, expansion chamber dust boxes and a multi-cone dust precipitator are provided in the suction line. The dust thus collected is returned with the mix.

The end point in the sintering operation is reached when all of the coal has been consumed. The mix proportion, feed rate, bed depth, and machine speed are adjusted; so that all of the added coal is burned out by the time the end of the machine is reached.

As the finished pallets of sinter break over the discharge end, the sinter cake is dropped onto an inclined plate which partially breaks up the cake and distributes it over a grizzly to remove the portion which is to be returned with the new feed. These fines are collected in a small bin to provide a uniform supply to the feeder. The grizzly oversize material is chuted direct to a railroad car for shipment. Since the sinter is still quite hot, sufficient water is directed to the outer edges of the load to cool this area and prevent car damage. The bulk of the sinter load is permitted to air cool to prevent thermal shock and spauling.

Because of the transportation and rehandling that are required to deliver the sinter to the blast furnaces, the sinter produced is of a "hard burned" quality to resist breakage. It is generally possible to deliver material which at the blast furnace bins will not contain over 5 pct of minus 20-mesh material.

The sintering operation alters the chemical composition of the concentrate slightly. The iron content is slightly reduced through the dilution by about 1 pct of ash from the mix coal. Fortunately, about 86 pct of the .21 sulphur contained in the concentrate is burned out along with the coal, resulting in a final sulphur content of about 0.030 pct.

Magnetite, Fe_3O_4, at elevated temperatures and under oxidizing conditions, is converted to hematite, Fe_2O_3. In the sintering of magnetite ores some hematite is always produced, the amount depending upon amount of fuel used and the degree of oxidation in sintering. On the average, we have found that we convert up to 14 pct of the magnetite to hematite; but we have found that it is possible to effect sufficient oxidation to produce enough hematite that the sinter cannot be picked up by a magnet.

Alternating Current vs. Direct Current

in Continuous Mining

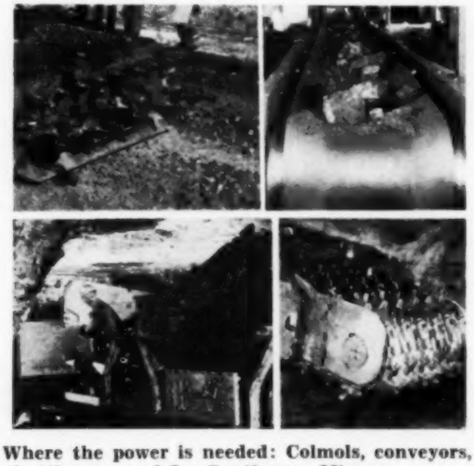
by J. R. Guard

Development of electrical power in coal mining has been an outstanding example of adaptability. It has accommodated itself to new inventions, changing mining methods, increasing demands, increasing safety requirements and many other new conditions.

With the appearance of the continuous miner in the mining machinery picture, those engineers interested in the application of electricity to mining have been thinking about what new combination of electrical apparatus will best suit this newcomer. It is the purpose of this paper to discuss the relative costs of supplying d-c versus a-c power to mining equipment with particular reference to continuous miner setups.

Assume that a-c power will be delivered to the mine at some high voltage suitable for transmission to the various substation locations. Typical voltage for this purpose is 4000 volts. The problem then is to change the power to some kind of power suitable for driving the motors of mining equipment. For safety and other reasons the voltage to be delivered to the motor terminals must be under 600 volts.

Motors on mining machinery, except traction equipment, may be suitable for either a-c or d-c. Traction equipment is generally unsuitable for a-c. The power problem then becomes one of changing the 4000 volts incoming a-c power to either low voltage a-c or d-c.



Where the power is needed: Colmols, conveyors, shuttle cars, and Joy Continuous Miners.

In order to compare the costs, assume a typical equipment setup including: 2 Continuous Miners, 4 shuttle cars, one 2000-ft conveyor belt with miscellaneous small motors for spraying and other requirements, all together requiring a substation capacity of 300 kw.

The cost of substation equipment suitable to change 300 kw of 4000 volts a-c power is as follows: $7000 for changing to a-c, $30,000 for changing to d-c by portable rectifier, or $17,000 by motor generator. There is a substantial saving in substation equipment costs with a-c regardless of the type of d-c conversion apparatus.

The efficiency of changing power to low voltage a-c is about 10 pct greater than that for d-c. Therefore, it will be necessary to purchase 10 pct more power if the job is done with d-c. This would amount to about .2 cents per ton for coal delivered from the conveyor belt of our example.

Mr. Guard is electrical and mechanical engineer for Rochester and Pittsburgh Coal Co., Indiana, Pennsylvania.

Whether the coal delivered by the conveyor belt of our example is hauled to the outside by locomotives supplied with d-c trolley power or by conveyor belts driven by a-c motors is a consideration beyond the scope of this paper. It is sufficient to say that foregoing figures apply equally to the power supply for this purpose.

There are additional advantages to using a-c. Generally speaking, a-c distribution systems in a mine are entirely separate from the haulage power and, therefore, not subject to interruptions which occur on it. Because of the absence of commutators on a-c motors, they are free from many troubles that cause d-c motor failures. Furthermore, the control devices for a-c are simple and less troublesome. The reliability of a-c motors and controls makes them much less costly to maintain. Transformers are nearly failure-proof as compared with d-c converting equipment. Fundamentaly a-c motors and controls are considerably cheaper than d-c equipment. For example, a standard open type 50 hp, 1750 rpm, d-c motor costs $819 against a cost of $559 for an a-c motor of the same rating.

With so many advantages in favor of a-c, the

natural question to be asked is why it has not been more widely used in mining. The answer is that it is not suitable for traction equipment. Another and less important reason is that a-c motors will not operate on poor voltage.

Looking back over the fifty years of development of electric power for mining, we see that haulage by trolley locomotives was the chief problem in the early days. Haulage was responsible for the largest part of the load and was probably the only load in some instances. It is not surprising, therefore, to find that the type power selected was that best suited to traction equipment. Furthermore, with the more dispersed systems of mining prevalent we had d-c power on the trolley wire throughout the mine. It was universally available at any active section in the mine. Because of its availability, d-c was used for driving practically all portable equipment and much stationary equipment in the mine. The outstanding exception was central pump stations that required large motors and freedom from power interruptions.

Besides having a universally available power supply, d-c motors have the decided advantage of being able to operate on poor voltage. It is true that under these conditions the speed is reduced and the motors may overheat, but in many cases it may be economical to continue the operation of a section with poor power temporarily rather than bear the expense of power improvement.

On the other hand, a-c motors will stall on reduced voltage, and it is necessary to maintain voltage within limits of 15 pct voltage variation.

Tests indicate that the peak power requirement for the Joy 3 JCM-C Continuous Miner will be about 150 kw. The average power requirement for a 20 second peak is about 115 kw. The average power requirement for continuous operation, including the lower power periods between the various portions of the cycle, will be about 105 kw. This means that a single machine, together with shuttle cars and spray water pumps connected with the machine, will require a substation capacity of 150 kw.

For the example used in this paper, consisting of 2 continuous miners, 4 shuttle cars, and a conveyor belt, a total substation capacity of 300 kw would be required.

This machine has a peculiar characteristic in that if the speed of the disintegrating chain drops, its torque requirement increases considerably, so that once it starts to slow down its power requirements go up rapidly. This means that speed should be maintained and that good voltage at the machine should be present at all times. A minimum voltage of 210 volts d-c would probably be allowable on 250-volt machines.

The present generally used method of transferring coal from the continuous miner to the conveyor belt involves the use of two shuttle cars per machine. For our example, this would then present a d-c power requirement of about 50 kw.

It is seen that power requirements for the miner are large and that poor voltage cannot be tolerated even with d-c motors. The quantity of power for a two-machine setup would take the full output of a 300 kw substation.

Since it is necessary to maintain good voltage at the motor terminals in any event, it would appear that the natural choice would be to equip the continuous miner with a-c motors and take the 4000 volt power up the entry to a point where the trailing cable from the miner could be attached directly to the secondary of a portable transformer. The transformer would be moved along the entry as work progressed, always keeping the transformer close to the work.

In addition to the transformer there would be a small motor generator set required to supply direct current to the shuttle cars. This set would probably be about 50 kw in size.

A second choice would be to locate a portable 300-kw motor generator set in place of the transformer and equip the miner with d-c motors. This would eliminate the necessity for a separate power unit for the shuttle cars. However, the m-g set would be much larger and more cumbersome to move along the heading.

The principal objection to locating the transformer or m-g set close to the work is that it requires the frequent moving of cumbersome substation equipment. If a-c motors are used it means moving the transformer and the small motor generator set.

A further and probably more serious problem is the matter of carrying the 4000-volt cable up the entry with means of shortening or lengthening it to follow the progress of the work. It is conceivable that this problem can be dealt with by using a large reel for the high voltage cable and carrying it on spool hangers.

It is to be noted that mining laws in some of the principal states forbid running high voltage cable in mines unless it is buried beneath the surface. The same laws forbid installing substations underground unless they are in approved type fireproof buildings. However, it is believed that if the safety and desirability of any system can be demonstrated, the laws can be modified accordingly.

If the substation is located at the bottom of the room entry and 250-volt d-c cables run up the entry to feed power to the machines, it will be found necessary to use rather large cables. Copper equivalent to 500,000 circular mills should be used in both positive and negative for each machine up to a distance of 2500 ft. For distances up to 5000 ft 1,000,000 circular mills positive and negative will be required.

For two Continuous Miners operating at 250 volts d-c and at a distance of 2500 ft, the cable cost alone would be $4600. This compares with a cost for cable of $1870 if power is transmitted at 4000 volts a-c.

To locate a transformer at the foot of the entry and carry low voltage a-c cable up to the point of use would seem an unlikely choice as a three-phase cable for this purpose would be too large to handle.

There is no doubt in my mind that complete a-c equipment is the correct ultimate application for the continuous miners, with 4000-volt portable transformers located near the machine. To make this choice more desirable some means, powered by a-c motors, must be devised to transport the coal from the continuous miner to the conveyor belt. Existing laws must be modified to permit the use of portable high voltage cables and transformers in the entries. When this has been done, the way will be clear for the full application of a-c in continuous miner sections.

Primary Blasting at Cananea

by K. R. Crocker

PRIMARY blasting at the open pit of the Cananea Consolidated Copper Co. has been developed to meet several conditions peculiar to the operation. Mining at Cananea is not confined to the open pit but is in conjunction with the underground mine which has been in operation for many years. Several old stopes included in the pit area were filled with waste obtained in stripping for the pit. The main haulage tunnel for the underground ore passes directly under one section of the pit and at present is but 180 ft below the pit floor. The pit as originally laid out called for an overall final pit limit slope of 60°. However, the upper 275 ft—a fairly soft oxidized capping, was carried at a 50° slope and the lower benches, which constitute 750 ft in combined height, are being mined to an over-all final slope of 60°.

These conditions make it imperative that the shocks produced by primary blasting be held to a minimum in order to prevent spauling from the backs of the underground drifts and to prevent cracking up of the final walls of the pit. The primary blasts are necessarily small, varying from 15,000 to 25,000 tons per blast. All haulage from the pit is by 30-ton diesel-powered trucks so the small blasts do not impair the loading and hauling efficiency.

Mr. Crocker, an AIME member, is open pit superintendent for the Cananea Consolidated Copper Co., Cananea, Sonora, Mexico. This paper was originally presented before the Arizona Section, Nov. 14, 1949.

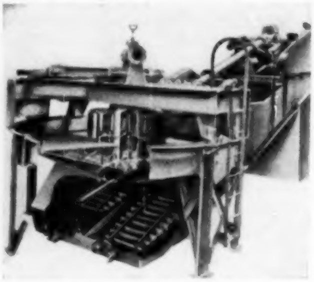
The rock is a monzonite porphyry which, in the south section of the pit, is brecciated and highly silicified. It is hard and dense and averages 11.5 cu ft per ton. Conventional standards for spacing blast holes in this type of rock resulted in very poor fragmentation and a consequently large amount of secondary blasting. As reported at the Arizona Section meeting in December 1948, the spacing of blastholes in this rock had been changed to provide a spacing between blastholes in a single row blast that is approximately 50 pct greater than the burden in front of the hole. Using a 17-ft burden and 23-ft spacing between holes, the fragmentation was greatly improved and the secondary drilling reduced to half of that necessary when using a 25-ft burden and 15-ft spacing. The powder factor remained at 3.5 tons per lb of powder. The wide spacing and short burden gives very little backbreak. In fact, the scars of the churn drill blastholes are visible after the blast to within 5 to 8 ft of the top on a 60-ft bench. Blastholes are drilled with a 9-in. bit to 5 ft below grade and loaded with 60 pct gelatin dynamite. For a 60-ft bench a single deck charge is placed 30 ft below the collar of the hole. A typical hole thus has a charge of 450 lb of gelatin in the bottom charge and 150 lb 30 ft from the collar. Two lines of reinforced Primacord are used in each hole, and extend to the bottoms of the holes.

The use of millisecond delay electric blasting caps at Cananea has further reduced the shocks caused by the primary blasts. As previously mentioned, the wider spacing of blast holes with less burden when fired in a single row with a single trunk line of Primacord gave greatly improved fragmentation. In using the millisecond delay caps at Cananea there has been no change in the spacing of holes but the holes are fired in such a way as to give in effect, a spacing of 46 ft. This is accomplished by using two trunk lines of Primacord. One trunk line is laid out in front of the holes and the Primacord from each alternate hole is tied to this trunk line. The intervening holes are tied to the second trunk line behind the row of holes. The front trunk line is detonated by two zero delay electric caps, one attached on each end of the line. The back trunk line is detonated by two 75-millisecond delay caps, one on each end. The fragmentation produced by this method shows a marked improvement. Spauling from the backs of the underground drifts has been almost completely eliminated. The detonation of alternate holes could be obtained by attaching the delay caps directly to each hole and connecting them to the electric circuit. However, the method described and in use is preferred at Cananea since it is felt that this method is safe from misfires due to a faulty cap.

A few experimental short blasts were fired using a double row of holes in order to produce a higher muck pile but in each case the backbreak was greater than desired although the fragmentation was fairly good. In our operation, where all haulage is by truck, there is no advantage gained by the double row blasting, and the elimination of backbreak is of prime importance to us —especially as the final pit limit of each bench is approached.

Dorr Announces the Hydroscillator, Copperhill Reports on Its Use

The Dorr Company has developed a new combined mechanical and hydraulic classifier, the **Dorrco Hydroscillator.** By a combination of upward-flowing hydraulic water and oscillation, mechanical mobility of the settled oversize is attained. The hydroscillator consists of two major parts—a shallow cylindrical oscillating classifying compartment, and directly below, a reciprocating rake dewatering compartment. A vertical motor drive unit imparts to the cylindrical oscillation compartment rapid circumferential oscillation about a vertical axis. Feed enters through a radial trough or launder terminating in an open-bottomed annular feed box at the center. Water under pressure is introduced into the bottom of the classifying compartment through perforations in an oscillating constriction plate. A "teeter bed" is thus set up, consisting of particles ranging from fines, close to the mesh of separation at top, to coarse granular particles at the bottom. Fine particles are swept outward by the radial flow of water and overflow across a peripheral weir. Coarse oversize particles flow to the periphery of the classifying compartment, pass under a stationary side wall and over an oscillating lip into the rake compartment. There they advance up the steeply inclined slope to the point of discharge.



Where the oversize particles rise in the annulus between the stationary side wall and the oscillating lip under the head imposed by the bed of teetering solids, they must overcome a back pressure set up by the head of water in the rake compartment, the water level of which is maintained at a higher elevation than the pulp level in the oscillating compartment. This results in a hydraulic balance which permits control of the fineness at which the separation is made.

A hydroscillator is in operation at Tennessee Copper Co., Copperhill, Tenn., and some results of the operation are mentioned on pages 709 and 710 of the article in this issue by J. F. Myers and F. M. Lewis.

SWECO h-m Plant Shipped to Philippines

A complete heavy-media separation plant left Los Angeles harbor last month, destined for the Philippines. Built by Southwestern Engineering Co. of Los Angeles, it is the first SWECO plant to be exported. Operated by one man, it differs from other heavy-media plants in that in a single separatory vessel it produces a middling product in addition to the sink and float products.

It will be used for concentrating chromite ore for the Consolidated Mining Co., whose property near Masinloc, Prov. of Zambales, Luzon, is one of the largest known bodies of chromite ore in the world. The ore is concentrated for shipment to the U. S., and used in the manufacture of refractory brick. The plant was designed to handle 4-in., plus 4-mesh ore, and to produce chromite concentrate containing more than 33% Cr_2O_3 and less than 5% silica. SWECO is planning to export more of these heavy-media separation plants, which are knocked down for shipment, and subject to simple field erection.

New Products

RockerShovels. After years of field testing The Eimco Corporation announces its new RockerShovel 104, which can be used for loading or bulldozing. Field tests have shown the 104 to be at least twice as fast as much larger conventional excavators that have many times the power, say the manufacturers. From ordinary sand and gravel to tough, abrasive rock excavation, the 104 is said to do a faster, cheaper job of loading. The Eimco 104 RockerShovel operates either backward or forward, with no swing around to discharge the bucket. The makers claim it replaces shovel, bucket and dragline excavating equipment at tremendous savings in investment and operating cost.

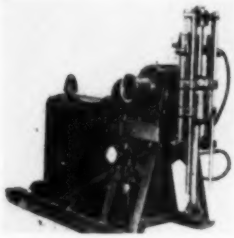
Compressors. Two new models of large reciprocating compressors—developed to meet commercial air conditioning and refrigeration application needs from 100 to 150 hp in single units and designed to operate with a variety of refrigerants—have been announced by the Carrier Corp., Syracuse, N. Y.

Concentrators. The Colorado Iron Works Co., of Denver, announces a new machine for gravity concentration—the Weinig concentrator. The machine consists of a cylindrical rotor operating on a vertical axis in a circular tank of somewhat larger diameter. A discharge opening in the center of the bottom and an annular sloping launder around the top provide for collection and discharge of sink and float products, respectively. This Weinig concentrator will be offered in combination with a dewatering unit consisting of a modified Akins classifier using a common pool level with the concentrator. The first major application of the simple machine will be on low-grade Mesabi intermediate ores.

Tri-Point Rock Drills for drilling granite, sandstone, hard limestone, concrete, etc., are now being made by Kennametal, Inc., Latrobe, Pa. Cutting tips are made of the vacuum sintered cement carbide manufactured exclusively by Kennametal, Inc. The triangular-shaped design of the thick carbide tips is said to be unique, and to give maximum resistance to wear and shock, as well as freedom from packing. Drill shanks are heat-treated alloy steel.

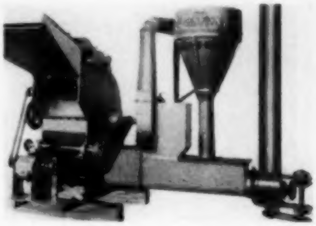
New Products

Vari-Purpose Hoses. Carlyle Rubber Co. of New York has made a new hose which they say will eliminate the necessity and expense of having several different types of hose on hand for various requirements. Vari-Purpose hose embodies the latest features in hose manufacture and claims following desirable characteristics: Specially compounded tube to resist many reactions of various elements, braided du Pont Cordura rayon carcass, maximum resistance to working pressures, oil and abrasive resistant long lasting cover, lightness in weight, extreme flexibility, and high serviceability at low unit cost.



Portable Core Drill. The Acker Drill Co., of Scranton, announces the new Acker "Teredo" core drill, designed to fill the need for a light, compact, portable core drill of medium capacity for depths to 600 ft.

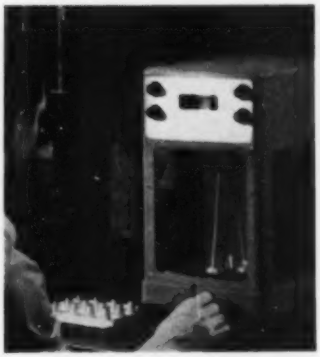
A new type of heavy-duty industrial hammer mill which features dustless operation has been announced by the Daffin Mfg. Co., Lancaster, Pa. Known as the Daffin Mill-U-Nit the mill is manufactured in two models, according to Irl A. Daffin, company president. One is the 12½-in., and the other 16½-in. intake. It grinds and pulverizes the most rugged materials, and operates well as a feed mill. Different sized screens are provided to change fineness of materials ground to desired requirements. Of all-welded steel frame construction, the Mill-U-Nit is designed for heavy ball bearings throughout with heavy ball bearing sthroughout with a steel roller chain to assure long wear. A special finger feed roll is



incorporated to handle roughage and fibrous materials, and a sectional hinged hood permits easy change of screens. A unique dust collecting system eliminates 95 pct of the dust. A screw conveyor is provided to carry finished materials to various heights to suit each user's needs.

Weighing ore samples to 1/20 mg with analytical balance accuracy, but in one-third the usual time is made possible by a new direct-reading instrument called the "Gram-atic Balance."

The new balance has only one pan, and the weight reads directly on a scale at eye level, eliminating a usual source of error in adding up individual weights. The balance has all required weights built-in and manipulated by turning four external knobs. No weights are handled, weights less than 0.1 g are indicated optically and automatically. Total weight is read directly from a scale on the instrument panel. The sample to be weighed



can range from 200 g (approx. 7 oz) to 0.0001 g.

A new protective plastic shunt for its Western electric blasting caps has been announced by F. S. Elfred, Jr., manager of the explosives division of Olin Industries, Inc. The new shunt is a sleeve made of a nonconductive plastic which is slipped over the bare ends of the leg wire, protecting them from corrosion.

A handoperated soil-sampling kit which can be carried in any automobile is being offered by the Acker Drill Co., Scranton, Pa. Designed for obtaining accurate, dependable subsurface information, the twelve sampling tools included in the kit can be used to obtain samples to depths of 25 ft in practically all soil and earth formations. Write to Acker Drill Co., Inc., 725 Lackawanna Ave., Scranton, for literature describing the kit.

For further information on new products mentioned here, write to the manufacturer, or to MINING ENGINEERING. When contacting manufacturers, tell them you "saw it in MINING ENGINEERING."

You Should Know That . . .

. . . **Joy trackless loaders** and shuttle cars are being used for development work in Southeast Missouri. Elmer A. Jones, assistant superintendent for St. Joe, describes advantages as speed of cleaning, ability to work on steep grades and sharp crosscuts, and good later tracking conditions (article, p. 679).

. . . **Equipment for determination of strain,** electrically, developed by the Baldwin Locomotive Works, Philadelphia, have found a completely new use as described by Louis Moyd on page 683. Although designed to measure mechanically-induced strains, the Corps of Engineers used them to determine coefficients of thermal expansion of mortar and aggregate for compatability tests in concrete. Petrographers and mineralogists might find the equipment useful.

. . . Another addition to the line of **electronic tube and equipment** applications to mining is the development of an electronic tramp iron detector for ore conveyor belts by C. M. Marquardt (see p. 703).

. . . Mill superintendents and manufacturers are following closely **studies on grinding.** On page 707 is a progress report on grinding with a large Hardinge mill being used at Tennessee Copper at low speeds. Relative wear rates of various diameter balls are expounded on page 712.

. . . Operating characteristics of the Colmol, one of two new **continuous coal mining machines,** are described on page 715. This equipment along with the Joy miner is receiving attention as a means of producing cheaper coal to meet competitive market conditions.

Belt Conveyor. The Hewitt-Robins sectional conveyor, which comes in short lengths designed for quick assembling, can be shipped as a package and installed in the field without the help of trained technicians in about one-fourth the time required for a conventional type conveyor. Aggregates can be conveyed at 300 tons per hr over the new conveyor which is primarily for small scale operations but it can also handle coal, coke, ores, and dirt. Hewitt-Robins Inc., New York, the manufacturer, will be glad to send information on this sectionalized conveyor.

Magnetic Tube Tester. The Dings-Davis magnetic tube tester, manufactured by Dings Magnetic Separator Co., Milwaukee, is available with an Alnico self-energized (permanent) magnet, requiring no electrical current. This unit is used to accurately and rapidly determine the magnetic content of iron ore which can be practically concentrated by the latest commercial wet type separators. It is not intended to show total iron content of an ore, but that which can be magnetically separated.

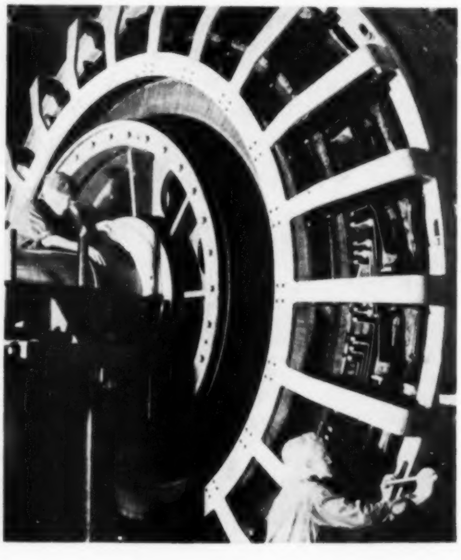
Largest Hoist for Kelley Shaft →

The largest mine hoist motor in the country from the standpoint of physical size has been shipped by General Electric Co. to the Anaconda Copper Mining Co. in Butte, Mont. This giant 600-volt d-c motor, rated 3000 hp at 60 rpm, will drive an ore hoist at the Kelley Shaft, an important part of Anaconda's $20,000,000 "Greater Butte Project" for the recovery of low-grade copper ore by the caving method. The G-E drive equipment also includes a 2500-kw, 600-volt d-c generator driven by a 3500-hp, 514-rpm, 2400-volt synchronous motor; a generator exciter; complete hoist control; and switchgear-type starting equipment for the MG set. G-E engineers state that the electric drive will be able to hoist 12 tons of ore per trip at a rate of almost half-a-mile a minute. Operating at full speed it will be able to complete a trip from a depth of 4335 ft in slightly more than two minutes. In the photo at right, G-E technicians are shown putting final touches to the motor before shipment to Butte.

Agitator Separator. The new Eriez nonelectric agitator separator is designed to separate small amounts of nonmagnetic material from large amounts of magnetic material. It consists of a permanent magnetic pulley, the surface of which is made of alternate lengthwise "slats" of Alnico alternated with slats of nonmagnetic material. A receiving chute or vibrator tray distributes the material to be separated to a nonmagnetic vane assembly from which it passes to the pulley belt, where the magnetic particles adhere to it until carried out of the magnetic field, at which point they drop into a receptacle. The non-magnetic material is not attracted to the pulley and drops through the vane assembly to its own receptacle. For complete information write Eriez Mfg. Co., State St., Erie, Pa.

Geiger Counter. Precision Radiation Instruments, Inc., Los Angeles, has designed its Model 106 portable Geiger Counter particularly for the prospector. It will detect beta particles as low as 160 K.E.V., as well as gamma, cosmic and x-rays, according to the manufacturer. It includes an earphone, a neon flasher, and a three-range meter as indicating means. A powerful electronic amplifier provides a loud audible signal.

A new line of super-rated V-belts, named "HY-T", incorporating a chemically produced fiber of extremely high-strength, low-stretch and excellent shock absorbing qualities, has been developed by the Goodyear Tire & Rubber Co. The Company claims that the new synthetic cord is also water and mildew resistant. The strength of the fibre is said to enable the belt to handle 40 pct more horsepower than standard belts without excessive stretch.



Development Work with Trackless Equipment

by Elmer A. Jones

Development work in mines of St. Joseph Lead Co., Southeast Missouri, using trackless loading equipment shows definite advantages: Speed of cleaning, ability to work on steep grades and sharp crosscuts, good later tracking conditions, and possibility of wasting rock in old abandoned stopes.

THE Desloge mine is one of a group of mines owned by the St. Joseph Lead Co., in the so-called Lead Belt of Southeast Missouri. It has been operating approximately 40 years producing a daily tonnage ranging from 1600 to 3800, depending on various factors. On an average, the present mine is 300 ft deep with an extreme extent of 2.7 miles by 1.7 miles.

Most of the present mining is in the lower horizons of the Bonne Terre dolomite, a fairly soft rock requiring no support except occasional channel iron and roof pins. Due to the contour of the underlying sandstone and porphyry, some development work has been through short stretches of these different formations. In some areas the lower part of the dolomite is quite sandy and generally is considered abrasive.

Sizes and Methods of Drifting Have Changed: Development work has, like other phases of mining, changed considerably in the last 20 years, from cleaning by hand to the use of a mechanical method. The conception of a standard drift has also changed. Before the advent of mechanical loading in stopes there were many pieces of main line drift averaging 7 ft high x 9 ft wide. With the coming of the St. Joe shovel the drift was increased to 8 ft high x 10 ft wide. Now with the Joy loader and shuttle car, a 9x11 ft drift is coming to be standard size.

Methods of cutting drifts have changed considerably over the last few years. Some changes in drilling equipment have been made such as reducing the weight of column, clamp, air feed, and fittings to go with the HC10 Cleveland jackhamer. Other changes have been made in drill steel such as the use of detachable bits; throw-away variety, screw type, and the tungsten-carbide insert bits. Later developments have introduced spiral-welded pipe with victaulic couplings, millisecond exploders, bulky powder, and the use of electric cap lamps. Experimental work is going on continually in various phases of development work, some to be tried and discarded, others gradually to become standard practice.

To sum up, the standard method of development now in use at Desloge is as follows: The work is done on a two-shift cycle 7:00 p.m. to 3:00 a.m. and 7:00 a.m. to 3:00 p.m. Two men known as development drillers comprise a crew on each shift. These two men arrive at their work about 7:30 to find a round blasted by the previous shift. After scaling down back and walls they proceed to the cleaning operation, using either a Joy loader and shuttle car, a drag hoist mounted on a slide ramp, or an Eimco Finlay to load into a 2½ ton mine car. Cleaning may take from 2 to 4 hr depending on conditions such as length of round broken, length of haul, and other variables. After fitting up pipe and setting up columns and drills they are ready to drill a round which should measure 9 ft high x 11 ft wide and break from 5 to 7 ft advance. This may also take from 2 to 3 hr. Then they are ready to charge and

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blast using 1x8 in. Gelamite Nos. 1 or 2 dynamite depending on the character of the rock. Failures to complete a round are not uncommon because of mechanical trouble, water, and various other reasons, but generally these crews will finish their swing.

Trackless Equipment Comes into the Picture: On January 1, 1948, the management made quite a momentous decision as far as Desloge was concerned. This was to shut down the mine as an ore producer and to go all out on development and remodeling. This program was designed to put development work a year ahead of mining and also to enable the ore to be hauled to Leadwood or Federal mill instead of Desloge.

This decision, increasing development work by about 100 pct over our wartime rate, called for more development men, more equipment, and more constant use of available machines. It was therefore necessary to split up old time development crews, taking on former operating drillers, and to plan on working as many crews as possible on a three-shift basis. Fortunately St. Joe had twelve Joy loaders and shuttle cars on order and Desloge mine was to get a total of five units. It was decided that they would be tried in drift work.

General Description Joy Loader and Shuttle Car: The Joy 18 HR is a continuous type loader with gathering arms in front head to throw the rock onto a universal chain conveyor (fig. 1). The material is discharged by moving the conveyor up or down and by swinging it 45° each way from center line to facilitate loading of shuttle car or mine car. The loader has a rated capacity of 5 tons per min with a maximum capacity of 10 tons per min. It is 66 in. high, 6 ft 4 in. wide, 25 ft long, and gathering arms will clean 5 ft 9 in. wide. The machine weighs about 15 tons, and is caterpillar mounted. The main driving motor is 75 hp. Two 7½ hp motors are mounted on conveyor frame to pull the conveyor. In addition there is a 5 hp hydraulic pump motor mounted in the rear to actuate swing of conveyor, raising and lowering head, and other motions.

The 60 E shuttle car weighs 12 tons and will carry about 13 tons of lead ore. It has a conveyor bottom for rapid discharge of load. The two 10 hp dc traction motors of the car can be powered by means of cable reel or trolley. The conveyor is operated by a separate motor of 10 hp. The car is 60 in. high and 25 ft long with a wheel base of 9 ft. The traveling speed is 6 mph empty and 4½ mph when loaded.

When it was known that Desloge would have Joy loaders and shuttle cars for future operating units, it was necessary to plan future development to accommodate these units. In other words we should enter an ore body from the side rather than the center and have our discharge point for shuttle cars about 4 ft above haulage way. One also should bear in mind that concentrating effort in a certain area would result in less time spent in long moves, closer supervision, maintenance, and so on.

By January 1, 1948, it was planned to cut 13,000 ft of drift during the current year. Thirteen development crews were organized, five of them on a three-shift basis, eventually to have Joy loaders and shuttle cars as regular equipment. The first loader and shuttle car, having been in operation, was put into a drift immediately using our men, who were experienced to some extent. The next four loaders and shuttle cars, called 18 HR and 60 E respectively, were delivered, a unit at a time, until by May 15, we were cutting five drifts using loaders and shuttle cars, on a three-shift basis.

Fig. 1—Shuttle car dumping into mine car.

Problems Presented Give Good Experience: The problem of training our drillers to use this equipment was readily apparent. Service representatives of the Joy Manufacturing Co. were available and were helpful in starting the training. The men on evening and night shift were, by force of circumstance, their own teachers, but found it easy to break in on the machines. In a few days each crew was doing a fairly good job of cleaning and hauling rock. Maintenance was a real problem as our repair men also had to learn the hard way. Some of the drifts being cut were 1 pct down-grade and as a result there was some water with which to contend. As long as the rock was hard and dolomitic the loading was fairly easy. In one drift a sugary sand was encountered. This sand disintegrated badly, and, mixed with water, was practically impossible to load as it never got back to the end of the loader. Here we had to substitute an Eimco Finlay loader and move the Joy to another drift.

Each drift had slightly different conditions. In the first one where a shuttle car was used, the waste rock was disposed of in an old worked out stope. Our first plan was to discharge the shuttle car on an up-grade making a higher and higher fill. We did not get far with this idea as the car would sink into the loose muck on the upper end of the fill and the flights going under the car would stick in this loose material. The remedy for this was to make a rock ramp on about a 10 pct grade ending at a log wall so that the rock could be piled up and later moved by a drag hoist and dipper to the desired spot. By the time this difficulty was overcome the develop-

ment crews had become so proficient in the use of the equipment that they were saving about an hour on their cleaning time and thus were able to do the additional work of spreading the waste rock on their shift. One immediate advantage of using the equipment was soon seen in this drift. In working three shifts the problem of ventilation was important. By saving time on the cleaning it was possible for each shift to have their round complete in about 6½ hr, giving 1½ hr for the smoke to clear before the next shift.

Another drift was started at an angle of about 60° from an old main line. The first 100 ft of this was cut using a hoist mounted on a slide ramp loading into mine cars. When the second loader and shuttle car were delivered the old loading equipment was taken out and replaced with the new machines. First we cut back up hill to the main line from a point about 30 ft in the drift and made a loading ramp about 4 ft above the level of the track. Due to a lack of sufficient small locomotives we had ordered Joy car pullers. One of these was installed on a bluff near the ramp and by using sheaves and moving the starting equipment to the ramp we were able to spot the cars under the end of the shuttle car without using more men. This crew then disposed of their waste rock directly into two ton mine cars and were dependent on main line car service.

A third different condition arose when it was necessary to cut a drift out of a stope on an upper level. A raise connected the two levels and it was possible to dump waste rock or ore into this raise from the shuttle car. This meant using a chute man to pull the rock out of the raise into mine cars and gave the development crews an opportunity to get rid of their rock.

After the first crew completed the drift we had another job that presented several new difficulties. A new lower level main line was planned to open up new ore bodies and provide haulage to Baker mine. It was to be 2400 ft long with several raises and crosscut drifts from the main line and about 40 ft lower than the old haulage level. It was possible to attack this job from three different locations. It also involved moving the loader, shuttle car, and drag hoist through about 3000 ft of old abandoned drifts, open stopes, and down an incline. This was accomplished by building a 440 ac power line as the loader moved ahead and installing 275 v dc feeder and return line as the shuttle car progressed. In some parts of the road there were small rock piles resulting from roof falls that had to be loaded and disposed of to clear the passage for the loader. The last difficult part of the trip was down an old bluff about 25 ft high. This had to be shot to decrease the slope from about 45° to about 30°. When the move was complete a drift from an old stope was started at right angles to the new main line location. This was only about 30 ft and then right angle turns were made to the left and right to start actual work on the new main line. This rock was disposed of in the old stope by using an Ingersoll-Rand 235 hoist pinned to solid, which piled the rock high in all directions. Very shortly after the drift to the left was started, porphyry was encountered. This was much harder than our ordinary dolomite and tungsten-carbide insert bits were used with good results. The condition continued for about 400 ft with no noticeable change in the efficiency of the drifting operation. By the time the drift broke through into part two of this haulage project the haul to dispose of waste rock was 950 ft. Each round was producing about 50 tons of rock which meant that the shuttle car had to make four trips to complete the cleaning job. The longest haul in the three sections of this total drift was 1250 ft.

Length of Haul: Since the use of the Joy loader and shuttle car in our development operations, about 15,000 ft of drift has been cut at the Desloge mine. Experience has been varied in determining economic hauling distances with one car. Several times the haul has exceeded 1000 ft. On two occasions, two trolley poles have been mounted on the shuttle cars, trolleys hung in the drift, and very satisfactory hauling results obtained. As the development crews become skilled it appears that a two-man drift crew can be expected to do their regular work in an 8-hr shift up to a distance of 1400 ft. In this maximum distance trolley wire should be installed to speed up the last 500 ft of the job. If the drift were to be longer than 1400 ft, we would stop and build track to a new discharge ramp located a safe distance back from the face. We have successfully done this job and are now well on the way toward completion of a drift 2000 ft long.

Track Work: Previously at Desloge and at present in other mines of the St. Joseph Lead Co., several different methods of tracking behind development crews were and are employed. Each method used required that trackmen be used at regular intervals. At Desloge there were four long main lines of development cut completely before any track was laid. Tracking here with 60 and 70 lb rails was done with the greatest efficiency we have experienced. Practically no digging was required for ties, perfect alignment was easily obtained, and exceptional speed was made in laying a finished track.

Ballast was obtained from digging out the necessary water ditches and cleaning up available loose rock. In the first drift we found that it was cleaned too well by the development crew and it was necessary to haul back into the drift sufficient fine rock to ballast the track completely. It was our experience in building main line through these clean drifts that the complete job of tracking was accomplished by one half the man shifts that would have been used tracking behind our old style, drift-cleaning equipment. The net result was also a much straighter and cleaner main line than could have been expected before.

General Experience: During the year and one half these machines have been in development including an interruption by a strike lasting 11 weeks, 15,000 ft of drift have been driven. Valuable experience has been gained both in discovering what the machines can do and what should be done to keep the machines operating.

Most of the main line work was on grades of 1 pct, both up and down. Occasional crosscuts were driven from the main lines, generally at sharp angles and on grades up to 10 pct. These grades and sharp curves were accomplished with no trouble either to the loader or car. In trying to load on one short crosscut that was 18 pct up, it was necessary to tie the car to the loader, a practice which is not recommended as feasible. On another occasion, a ramp to discharge rock into a pile was built on a 15 pct grade for a length of about 80 ft with a flattened out section at the end. This job was successfully carried out with trolley wires running up this grade. Turns in crosscuts as sharp as 25 ft radius were also made without difficulty. The only serious trouble encountered due to physical conditions was in trying to

load disintegrated sand in water which is virtually impossible to accomplish.

Another advantage soon evident was in speed of cleaning. In our method of drifting, a center cut is used which throws rock back down the drift as far as 300 ft. Other machines work very slowly in thin fly rock but these loaders and shuttle cars are very fast in this kind of loading. It is possible to bulldoze long distances in thin rock and, when ready to load, the shuttle car can be made available. Superficial time studies and comparisons indicate a saving of about an hour on an average swing of drift. As soon as this was evident most of the crews immediately started drilling a longer swing of drift. A 7-ft average for each round over a week's work was not uncommon and 6-ft rounds were the rule rather than the exception. These average rounds compare with drift rounds in the rest of the district of $4\frac{1}{2}$ to 5 ft. The greater speed of loading is attributed to: (1) ability to load fly rock faster, (2) ability to load in good rock at a rate of 90 tons per hr, and (3) a shorter period over all the cycle to wait for disposal of rock after it is loaded.

Fig. 2—Typical stope loading operation.

Power: Our loaders were ordered, due to local conditions, to run on 220 v ac. Each unit was accompanied by one 50 kva dry transformer with 440-220 taps. We found that with our fluctuations in voltage, it was best to keep the transformer run up within 800 ft of the shovel. Current on the 440 v side was supplied through 2-0 or 300,000 c.m. rubber-covered wire.

The shuttle car, operating on 275 v dc is limited to certain conditions. Our motor generator sets are placed around main line haulages about 4000 ft apart. Much of this development work was far off present main line track so that the problem of sufficient power and return lines was almost a constant one. However, by using 4-0 trolley or 300,000 c.m. feeder line with 300,000 c.m. return to a railroad track, it was possible to use the shuttle car in any job. With normal demands on motor-generator sets it is probable that an economic limit of about 3000 ft is practical. The wisest plan is to check voltage on both machines and not allow too great a voltage drop on either of them.

Car Spotters Aid in Unloading: Car puller proved to be a necessary adjunct to successful operation of the shuttle car when it is discharging into mine cars (fig. 2). Spotting cars under the end of a shuttle car with a locomotive is a slow, uncertain job, particularly with a long train. It also requires an extra man on the crew to handle the locomotive. By using a car puller and moving the train up-hill under tension, it was easy to have the car always in the right spot. We also rigged up starter buttons within reach of the shuttle car operator so that he could control the movements of the car puller from his station. By using the car puller, we were able to save an extra man and do a quicker job of spotting cars.

Maintenance: Maintenance of machinery in any development operation is a problem. With trackless equipment it should be realized that this is a complicated machine, heavy in construction and expensive. It is not news to mine operators that good development men are usually poor maintenance men. However, equipment of this nature requires care on the part of the operator and also good maintenance to prevent the machine from wearing out and losing production by having breakdowns.

Our experience has been with men, untrained mostly, in operating and maintaining this equipment. Although now they are much better than they were, they still lack certain fundamental knowledge of the machine and also the value of preventative maintenance. For full value and lowest possible repair costs, considerations should be given to picking the right man as an operator and to giving him proper training in the operation of the machine, the working parts, and the correct lubrication of the equipment. Then, if this kind of man is available, provide enough supervision to be sure he does what he is supposed to do.

Conclusion: Management of St. Joseph Lead Co. bought this equipment with the idea that it would replace the old Thew shovel which had outlived its usefulness due to the fact that ore bodies were becoming smaller and lower in height. It was hoped that it would supplement the St. Joe shovel by being more flexible. This proved to be the case when it was successfully used in trackless development work. Whether or not it could be purchased and used only in development work and prove more economical than the Eimco, Conway, or drag-hoist is still more or less problematical. Having had this experience with trackless equipment, we find that we have a unit that is very flexible. It can be used successfully in stope operations, low ground, and narrow, twisting and turning ore bodies, and also in drift work in all its phases peculiar to St. Joseph Lead Co. operations.

Determination of the Coefficient of Linear Thermal Expansion of Rock Specimens by Means of Resistance Wire (SR-4) Strain Gauges

by Louis Moyd

The Concrete Research Division, U.S. Corps of Engineers, has developed a simple procedure for determining the coefficients of linear thermal expansion of rocks by means of resistance wire (SR-4) strain gauges. Gauges are cemented to the surfaces of specimens, which are then brought alternately to temperatures of 35°F and 135°F. Results are accurate to the seventh decimal place.

IN the course of investigations of the thermal properties of various rocks proposed for use as aggregates in concrete structures, a relatively rapid method of determining coefficients of linear thermal expansion was required. A method which proved to be satisfactory for this purpose was developed in the petrographic laboratory of the Concrete Research Division, U.S. Corps of Engineers, at Clinton, Miss. The method is based on the fact that strain effects in a specimen can be determined by measuring variations in the resistance of an electrical conductor firmly cemented to the surface of the specimen.

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Equipment for the determination of strain, electrically, has been developed by the Baldwin-Southwark Division of the Baldwin Locomotive Works in Philadelphia. This equipment includes small strain gauges made of coils of resistance wire bonded between thin sheets of paper (SR-4 gauges), and a portable "Strain Indicator" which records variations in the length of the gauges in microinches per inch. Gauges of this type are now in general use for determinations of various mechanically-induced strains, but their use for the determination of coefficients of thermal expansion appears to be a new application. The advantages of the method are that the test specimens may be relatively small and require only a minor amount of preparation, that a large number of tests can be run at the same time, and that the installation and operation of the equipment is relatively simple. The method and equipment described are now in regular use for determining the coefficients of linear thermal expansion of rock samples submitted to this laboratory. Rocks which have been tested include granites, syenites, gabbros, and other igneous rocks, limestones and dolomitic limestones, sandstones, and quartzites.

Specimen Preparation: To standardize the conditions of the tests, the rock specimens are sliced by diamond saw into slabs about 1½x1¼ in. In the case of bedded or foliated rocks, three oriented slabs are cut from the same specimen, and represent, respectively, a direction in a plane of the bedding or foliation of the rock; a direction along the bedding or foliation and at a right angle to the first; and a direction across the planes of bedding or foliation. One of the large surfaces of each slab is lapped for a few minutes with No. 100 abrasive powder to remove saw scars and irregularities. The slabs are soaked in petroleum ether and then dried in an oven to remove all traces of oil picked up in the sawing and lapping operations. When the specimens are dry, two SR-4 gauges (with active gauge-lengths of ¾ in.) are cemented to the lapped surface of each slab with Duco cement. All bubbles are pressed out, and weights are placed on the gauges (with an intervening layer of blotting paper) and left on overnight while the cement hardens and cures.

Instrumentation: When the cement has cured, the slabs are placed on a panel which can hold up to 36 specimens, and the strain gauges are connected into the test circuits by means of binding posts. Panels made of Transite and of Lucite are now in use. The circuits, as presently arranged, go through two eleven-point, three-bank, silver-contact selector switches. Each gauge connection is coded so that its reference points on the switches are known, and the gauges are read in the same sequence each time. The portable strain indicator is a completely self-contained unit including batteries, and is primarily a Wheatstone bridge, with additional components permitting dial calibrations in microinches per inch, and an adjustment for the use of gauges of different lengths and initial resistances. To minimize temperature effects on the gauges themselves, the circuit requires that a mounted dummy or compensating gauge be balanced against the active gauges. After an extensive series of tests, it was found that the best results were obtained when the compensating gauge was mounted on a slab of material with a known (and relatively large) coefficient of linear thermal expansion, and kept in the same environment as the specimens being tested. In practice, the compensating gauge is mounted on a slab cut from a quartz crystal, with the gauge direction parallel to the C-axis of the crystal. A test gauge is mounted parallel to the compensating gauge on the same slab. As a check on the quality of the test results, slabs having known coefficients of linear thermal expansion are interspersed among the test specimens. In practice, an oriented slab of quartz and one of fluorite are used. The panel containing the test slabs is placed in a cabinet, the humidity and temperature of which can be controlled. Three thermocouples are attached to a slab similar in size and shape to the test slabs, one of the thermocouples to the upper surface, one to the lower surface, and one cemented into a hole drilled longitudinally through the slab. The thermocouples are connected to a recording thermometer outside the cabinet. It was determined that the 100° temperature range between 35° and 135°F was most significant for concrete aggregates, since effects of freezing are introduced into concrete below that range, and temperatures above that range are rarely, if ever, encountered even on exposed surfaces in the warmest climates. The portable strain indicator is placed outside the cabinet and connected to the test panel by a multistrand telephone-type cable. A simplified wiring diagram of the test circuit is shown in fig. 1; the equipment used is shown in figs. 2 and 3.

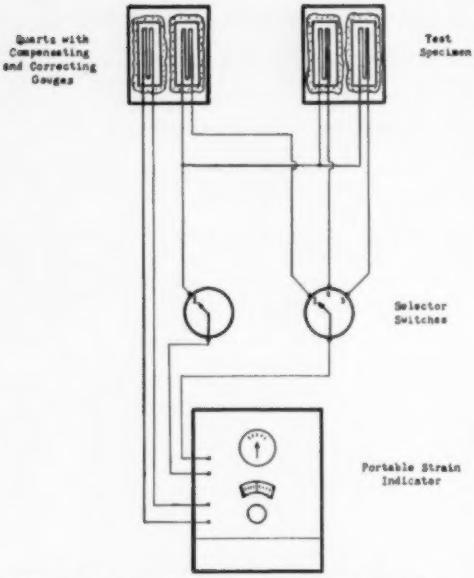

Fig. 1—Wiring diagram for determination of coefficients of linear thermal expansion by use of resistance wire (SR-4) strain gauges.

Procedure: After the cement holding the gauges to the test slabs has been sufficiently cured, and all the gauges are connected into the test circuit, the panel is placed within the controlled-humidity and temperature cabinet. The humidity is maintained at a minimum, and the temperature is raised to 135°F, then reduced to 35°F. No readings are taken during this first cycle, since it has been found that very erratic results are obtained until the slabs and mounted gauges have been flexed at least once. The temperature of the cabinet is then varied from 35° to 135°F, alternately, for a series of ten complete cycles, and the gauges are read at both extremes of temperature. The temperature of the test slabs is considered to be uniform throughout when all three thermocouples reach the same temperature. It was found, however, that gauge readings were more consistent if they were taken after the slabs were allowed to remain at the peak temperature for about half an hour. This condition was not found at the lower temperature. With the equipment used, it is possible to obtain readings covering two full cycles per 8-hr day, thus each complete test takes six days including one day for preparation of test specimens.

Computations: All of the gauge readings are expressed in microinches (1 in. $\times 10^{-6} = 0.000001$ in.) per inch. A correction factor for each run is based on the difference in the readings at the maximum and minimum temperature of the gauge attached to the same quartz slab as the compensating gauge. Theoretically, there should be no difference in these readings, since the resistance of the compensating gauge should completely balance the resistance of the test gauge. Use of this correction factor in the computation of the coefficients of linear thermal expansion of the test specimens results in good determinations for those specimens with known coefficients, and therefore this factor is a necessary part of the computations. It is possible that the correction eliminates errors introduced into the system through extraneous forces, such as variations in room temperature, which might affect resistances in lead-in cable, switches, and portable indicator. In computing the coefficient for each test specimen, results of both gauges are averaged, unless one has given erratic readings, in which case it is presumed that the gauge or the wiring is faulty and the readings of this gauge are disregarded. A typical computation of the coefficient of linear thermal expansion for a test specimen, based on SR-4 gauge readings of a run between 35° and 135°F, is shown below:

Coefficient of quartz, upon which compensating gauge is mounted, is taken at 4.30×10^{-6}. This multiplied by the 100° temperature difference equals 430×10^{-6}, which is the compensation factor.

Reading of active gauge on quartz, at 35°F (attached next to compensating gauge)	$1{,}435 \times 10^{-6}$
Reading of active gauge on quartz, at 135°F	1,395
Difference = Correction Factor	—40
Reading of gauge on specimen, at 135°F	1,060
Reading of gauge on specimen, at 35°F	—965
Apparent expansion of specimen	95
Compensation factor	430
Correction factor	+40
Total factor	470
Total factor	470
Apparent expansion of specimen	+95
True expansion of specimen	565

True expansion of specimen, 565×10^{-6} per in. divided by 100° temperature range equals 5.65×10^{-6} in. per in., which is the coefficient of linear thermal expansion for the specimen.

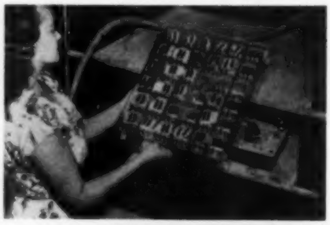

Fig. 2—Resistance wire (SR-4) strain gauges cemented to rock slices and connected into circuit through binding posts on Lucite test panel.

Fig. 3—Equipment used in determination of coefficients of linear thermal expansion of rock slices, including controlled-temperature and humidity cabinet, multiple-strand cable, switch panel, portable strain indicator, and recording thermometer.

This is the result of only one run (½ cycle). The averaged results for the complete 10 cycle test (20 runs up and down the temperature range), always give good determinations (accurate in the first decimal place) for the interspersed, known specimens, and therefore it is considered that acceptable results have been obtained for all of the rocks being tested.

Acknowledgments: The application of resistance wire strain gauges for the determination of the coefficients of linear thermal expansion was conceived by the late Elliott P. Rexford, Geologist, Los Angeles District, U.S. Corps of Engineers, who used the gauges to investigate the thermal expansion compatibility of mortar and coarse aggregate in several large samples of concrete. Albert Weiner, Engineer, North Atlantic Division, U.S. Corps of Engineers, assisted in the development of the procedure in present use.

Deposits of Heavy Minerals on the Brazilian Coast

by Joseph L. Gillson

BRAZIL has had an industry based on ocean beach deposits of heavy minerals containing monazite, zircon, rutile, and ilmenite for well over 40 years, but except at the very earliest period, prior to 1906, and again during World War II, has this industry been at all comparable in size with similar operations in other countries. Limiting factors have been neither a lack of reserves nor lack of markets (except for zircon). Although poor local transportation and shipping facilities have handicapped the development in Brazil, these conditions were perhaps equally bad (when mining operations started) in the Indian State of Travancore, and in New South Wales and Queensland, Australia, in which states the principal production of beach deposits of heavy minerals has occurred. The outstanding restraints on the Brazilian industry have been the lower quality of the Brazilian minerals, the relative small size of individual deposits that were known until recently, the lack of interest of Brazilian investors in mining ventures, and the unfavorable climate for investment of foreign capital under Brazilian mining law. The lower quality of the minerals is indicated by the fact that the Brazilian monazite carries only 5 to 6 pct ThO_2*, and the ilmenite 56½ pct TiO_2, whereas the corresponding Travancore minerals carry 9 pct ThO_2 and 60 pct TiO_2 respectively. The Brazilian zircon is a dirty brown color (which can be removed readily) as compared with the gleaming whiteness of the Australian zircon. An inadequate market for the abundant zircon that will be available in any operation in Brazil is certainly a heavy restraint, since otherwise it could be an important byproduct. The oxide of zirconia, baddeleyite, that has come from Brazil is from other sources.[1] Since the Brazilian beach sands carry only a little rutile, that mineral has not been available as a valuable byproduct to help finance the operations.

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* This was important in the days when the manufacture of gas mantles was the only outlet for the chemicals made from monazite. Today, since the other rare earths in monazite are used in industry more than thorium, the Brazilian monazite is of equal value to the Indian.

Substantial tonnages of rutile have been exported from Brazil. The sources of this mineral are stream placers and residual deposits over gneisses in the States of Goyaz, Minas Geraes, Ceara, etc. (Leonardos,[2] and Chambers[3]).

With these handicaps, Brazilian production from beach sands has been marginal, and there has been only one consistent producer. This is a company, originally French, which was called Companie de Franco-Brazilliéne, but, with the downfall of France in 1940, became Brazilian under the name Monazita e Ilmenita do Brasil ("Mibra"). This company has operated several deposits in the vicinity of Guarapary, in the State of Espirito Santo. Its office and plant has been at Guarapary.

Recently a company called "Fomil" (Fomento Monazita-Ilmenita) started operations. This company was organized by the Foote Mineral Co. of Philadelphia, and such share as a foreign company can hold under Brazilian law has now been acquired by the Lindsay Light and Chemical Co. of Chicago.

When the writer went to Brazil in the fall of 1940, he found that no general survey of the beaches had been made except for a thorough study of part of the area by Abreu.[4] The paper by Miranda[5] had not then been written. The federal Geological Survey and Department of Mines knew only of the existence of the deposits in Espirito Santo south of Vitoria, and had a hazy idea that others might be found north of Vitoria. One deposit in Baía was known in Rio, but the larger one at Guaratiba was not known, although a Frenchman had prospected

it rather thoroughly 20 years before. No knowledge of the deposits in Natal existed.

The discovery of extensive and numerous deposits resulted from visits to areas favorable geologically and there making local inquiries for knowledge of black sands, and also "wild catting" potential beaches. Reserves in deposits newly found exceed the sum of all those known previously.

Throughout this article the black titanium mineral is called by its commercial appellation, ilmenite, although mineralogically it is arizonite except in two deposits. Arizonite is the ferric iron analogue of ilmenite, and is higher in TiO_2 (ref. 6, p. 1042). In a recent paper by Overholt, Vaux, and Rodda,[15] an opinion is expressed that arizonite is weathered ilmenite. The writer questions this generalization as not being in accord with world occurrences.

Location of the Deposits

The Brazilian deposits of heavy minerals on ocean beaches occur in a zone about 175 km long, extending north from the northeast corner of the State of Rio de Janeiro, up into the contiguous State of Espirito Santo as far as the mouth of the Rio Dôce. Further north there is a short zone in the southern part of the State of Baía, about 40 km long. No other deposits are known except in the States of Paraíba do Norte and Natal on the "hump" of Brazil (figs. 1, 2, and 3).

This limitation of the deposits to specific zones has a direct and positive relation to the local geology. The deposits occur where the bottom beds of the coastal plain are being eroded, or were recently eroded, by the sea.

Fig. 1—Index map of Brazil.

The sections of the coast in the States of Rio de Janeiro and Espirito Santo are moderately well populated and developed, but they lack roads parallel to and near the coast, and there are only a few ports, only one of which can be entered by large ships and at which there are docks and facilities for handling cargo. This is the port of Vitoria, the capital of the State of Espirito Santo. Only some of the smaller deposits are near enough to Vitoria to permit consideration of trucking the products from the mine to the dock. The small port of Guarapary, about 50 km south of Vitoria, can be used by vessels up to 3000 tons, but port facilities are inadequate. Other ports along this section of coast are so small that only lighters and small fishing boats can enter.

An anchorage north of Itapemirim (fig. 2) can be considered as a potential loading point for the big deposits in the north end of the State of Rio de Janeiro, but roads south from the anchorage are nonexistent below Marataízes.

The deposits in the southern part of the State of Baía can be reached from the small port of Caravellas but roads serving the coastal area are very poor. Offshore loading from these beaches is possible, however, because of the protection afforded by reefs, and before 1905 there was extensive shipping off these beaches.

The deposits in the northeast coast have been inadequately explored. Those near the mouth of the Paraíba River below São João de Passôa are reasonably accessible, but probably too low grade, while others in Natal, near the fishing village of Cunhaú, are large but poorly located for shipping.

Types of Deposits and General Geological Features

A very large area in east central Brazil is underlain by pre-Cambrian gneisses.[7] The coastal belt of gneisses extends north from the State of Santa Catharina, with but one interruption, for 2000 miles into the north coastal section in the State of Ceara. Drs. de Oliveira and Leonardos[7] have shown that the belt is nearly 300 miles wide in the vicinity of Rio de Janeiro, 150 miles wide in Baía, and nearly 400 miles wide in Pernambuco and Paraíba do Norte.

The gneisses reach the coast from the southern limit of the pre-Cambrian area as far north as Cabo Frio, east of Rio, and the familiar Sugarloaf Mountains of Rio harbor are spectacular physical features of the gneissic terrain, and owe their form to exfoliation and cleavage along the planes of structure.

From Cabo Frio north a short distance to the town of Macaé, the gneissic cliffs continue to the sea, but from there north almost without interruption, there is a fringe of coastal plain sediments along the actual coast (figs. 3 and 4). For long distances, however, the coastal sediments are so thin that the underlying gneisses are seen at the water's edge or in islands offshore. The delta of the big Rio Dôce (fig. 2) buried the older coastal plain sediments in the vicinity of Regencia, and from there north to Caravellas the shore is cut in a higher section of the Tertiary beds. For a short stretch near Caravellas the lower section is exposed, and again deposits are found. Higher and higher beds are found at the coast to the north, and the stratigraphic section is so thick that it has been explored for oil in the vicinity of São Salvador (fig. 4).

In Paraíba do Norte and in southern Natal, the coast is cut in Cretaceous beds, which also carry a little ilmenite.

The geographic association of the beach deposits of heavy minerals with the bottom beds of the

coastal plain is too consistent to be coincidental. The formation of the deposits of heavy minerals is related so intimately with the shoreline developments that a review of the recent physiographic history is necessary.

The gneissic area has had at least two periods of peneplanation. They may correspond in general age with the Cretaceous and Tertiary peneplanes of the Appalachian region.

Only in the soil overlying the Cretaceous land surface was there a concentration, by extraction of soluble minerals, of the grains resistant to chemical decay. Such minerals are ilmenite, rutile, zircon, staurolite, sillimanite, kyanite, spinel, tourmaline, and others. Magnetite, which occurs in the unaltered gneisses with the ilmenite, had been almost completely eliminated by weathering. This important elimination of an objectionable associate of ilmenite before transport of the products of rock disintegration to the coast is an essential step in the later formation of beach deposits of high-grade ilmenite. Similar conditions had obtained in India, Ceylon, and other places where high-grade deposits of ilmenite occur. On the other hand, where a black sand is found in deposits formed only by mechanical disintegration of the parent rock, the ilmenite is contaminated with magnetite from which it cannot be separated completely. Such a deposit of titaniferous magnetite is found in Brazil in the delta of the Rio Dôce which is made up of material brought down from the deeply dissected valley recently carved in the hard rock.

The source of the abundant monazite can be inferred to be from pegmatites which are widely distributed in the gneisses. This inference is made simply because monazite is known to be an accessory in many pegmatites in North America.

Following the Cretaceous peneplanation, uplift caused rejuvenation of the drainage, and the deep surface soil containing these resistant minerals was removed by rapid erosion and dumped into a shallow sea lying over the continental shelf, where the sand was deposited at once and the accompanying clay washed away. Thus, the basal beds of the coastal plain sediments carry everywhere a small percentage of these resistant minerals. In no place has the content of these minerals been found to be sufficient to justify consideration of mining the Tertiary rock as such. Had not subsequent marine erosion of these beds occurred with further concentration of the minerals by waves and currents, there would be no deposits of workable grade in Brazil.

The more recent history of the Brazilian coast has been marked by repeated changes in level of the land with respect to the ocean. There has been tremendous faulting parallel to the coast, which has dropped the east side and left inland, elevated and undisturbed, the old-age erosion surfaces that now have no connection whatsoever with the precipitous cliffs and land surface in early maturity, characteristic of the down-dropped coastal section. In no place is this contrast seen more clearly than in the automobile drive from São Paulo to Rio de Janeiro. The city of São Paulo is spread out on an old land surface along a big river, Paraíba do Sul, which flows tranquilly north and parallel to the coast on a gentle grade, oblivious of the tremendous falls and canyons ahead. Upon reaching the "Serra" near Rio, one looks back on this land surface of old age, but forward down toward the sea over a tremendous fault scarp, up which the roads twist and wind in breath-taking curves, and down which the youthful

Fig. 2—Map of a part of Estado do Espirito Santo showing ilmenite deposits.

streams rush headlong to the sea, 1000 m below. At other points, a puffing railroad engine pushes only one light car at a time up the cogwheel roads from Santos to São Paulo, and from Rio to Petropolis and Rio to Terezopolis, climbing up this eroded fault scarp.

After the present topographic form of the coastal area by deep dissection of the fault scarp was well advanced, drowning occurred, and the deep harbors of Rio and Vitoria resulted from the submergence of the mouths of old rivers, and of low-lying areas. By submergence, the harbor at Rio, the "most beautiful harbor in the world," came into being, and also every stream along the coast became marked by a wide estuary at its mouth. This drowning, comparable in degree and probably in age with that of the North American continent which produced the

Fig. 3—Brazilian coast from Anchieta to Rio de Janeiro showing ilmenite deposits at Barra do Itabapoana and Bôa Vista.

Fig. 4—Geological map of Brazil.

estuaries of the Chesapeake, Delaware, and Hudson, was sufficiently remote in time (probably early Pleistocene) for subsequent advanced development of the coast line. Many of the more pronounced headlands have been lopped off, and deeply indented bays were closed off by sandbars, behind which salt marshes developed which gradually filled.

Very recently geologically, there has been a modest uplift of about 5 m, and textbook examples of all of the features characteristic of recent emergence are visible along the Brazilian coast. Old sea-cut cliffs stand back from the sea, with old shore lines buried under a mass of thick vegetation. Elevated offshore bars stand now "high and dry" as ridges between the present shore and the old sea cliffs. The estuaries at the mouths of smaller streams are closed by elevated bars behind which are lagoons.

Where the pre-Cambrian gneisses come down to the sea, the shore line prior to this emergence had been very rugged (as at Rio). The slight emergence has put a strand line around the base of the old cliffs. Except for this emergence and consequent formation of a strand line, there would have been little flat land on which to build a city such as Rio. The popular suburb of Rio, called Ipanema, is built on the elevated bar built across an old bay. This is now closed off from the sea to form the lake called Lagôa Freitas.

Where the more easily eroded coastal-plain sediments lie at the coast north of Macaé, the modification of the shore line, after submergence, had been further advanced than in the harder and more resistant crystalline rocks. A particularly well-defined sea cliff had formed by the rapid erosion of the soft sediments by the waves. An earlier terrace had been established on these soft sediments, probably by stream benching, and the height of this terrace at a particular place depends on the distance the old valley lay from the seacoast prior to submergence. The sea cliff formed a scarp below this terrace which, in many places in the belt where the black sand deposits occur, is 20 to 50 m high. The old elevated sea cliff is such a conspicuous part of the Brazilian coastal landscape, that it is known generally by the residents as the "barreira" (barrier or cliff).

Since the emergence, in places marine erosion has been sufficient to cause the complete destruction of the elevated strand at the base of the scarp, and attack on the old sea cliff has been renewed.

Prior to the emergence, waves were breaking against the barreira and offshore bars were developing from headland to headland in the familiar process of shore-line straightening. A sketch attributed to Douglas Johnson[8] showing "submaturity" in shore-line development of submergence, represents most accurately (if too simplified) the coast of Brazil in the State of Espirito Santo. If the effect of slight emergence were added to that picture, and a strand line were drawn in front of the old cliffs and bars extending across the bays, then the present shore line would be indicated.

Prior to the emergence, the waves dashing against the sea cliff undercut it, causing large chunks of the poorly consolidated material, of which the Tertiary sediments are made, to fall into the waves from time to time. The waves digested these chunks, carrying the minerals of light weight away, and leaving the heavy minerals on the beach about where they fell. Thus, a black beach formed at the base of the barreira; black, since the most abundant of these heavy and resistant minerals is ilmenite. Such concentrations of heavy minerals are forming today as marine placers at Maíba in Espirito Santo, and at Comoxatiba in Baía. Elevation of these beaches left at the foot of the barreira a black sand deposit now thickly covered with brush, or used for the cultivation of mandioca, as at Saí and Ponta da Fruta.

Prior to the recent emergence, enrichment of the sand on the offshore sand bars occurred in a manner very similar to the action of concentration on a Wilfley table. The sand derived from the destruction of a headland was moved along the coast under the influence of along-shore currents. A little concentration took place during this migration since the grains of lowest weight were lifted by the restless motion of the water and remained in suspension. As the bar grew in size and height, it began to give resistance to the incoming swells from the ocean, and breakers formed. As these "rollers" dragged bottom on the bars, they caused a movement crosswise with the length of the bar, and this movement had sufficient force to carry the lightweight minerals off the bar into the deeper water on the landward side. These rollers thus acted like the wash water on a Wilfley table. The heavy minerals remained behind on the bar, thus the sand was enriched by subtraction of those minerals of light gravity.

For a while the bars were beaches themselves, but, with the progressive elevation of the coast, emergence lifted them above the waves and permitted the wind to do some work. In most places the wind dumped some barren sand on the elevated bars so that the top layer may be quite barren. Continued emergence exposed a strand line on the sea side of the elevated bars. This strand is now the actual shore and in most places contains no "heavy mineral" of importance.

There are thus the following types of deposits of heavy minerals on the Brazilian coast: (1) Elevated beaches. (2) Elevated bars. (3) Modern beaches. (4) Dune deposits formed by the action of the wind on (1), (2), and (3). Since dunes are not a conspicuous feature of the Brazilian coast, presumably because of the rapid growth of protective vegetation, deposits of this kind are rare. The only one important in size and grade is at Cunhaú in Natal.

(5) A delta deposit at the mouth of a big river which transported material derived from recent deep-river erosion into fresh rock.

Comparison with Similar Deposits Elsewhere

The relative small size of the individual Brazilian deposits as compared with those in Travancore, India, results presumably from the fact that the "protore,"* that is, the sand in the coastal plain sediments, was itself so low grade that the quantity of mineral available in one place was only of modest proportions, whereas locally in Travancore, the "terra rosa" which formed during tropical weathering as a deep soil zone over the gneisses, was considerably enriched in heavy minerals. In Travancore, black sand lies in every roadside ditch cut in the red earth at Manavalakurichi, at Quilon, and along the highway to Alleppey.[8] In Brazil, only traces of black minerals are visible in drainage channels around Vitoria or Caravellas. Hence, in Brazil, deposits formed only where the physical conditions of wave, wind, and ocean current were most favorable to concentrate the moderate amount of black sand available. In Brazil, however, the protore was available for scores of kilometers, and numerous deposits resulted. In Travancore, the low-lying shore did not offer general conditions for rapid marine erosion. The heavy mineral was brought down to the sea by big rivers, and the two deposits on the Travancore coast lie only where the mouths of these rivers were at one time. In Brazil, except for case 5, rivers had little or nothing to do with transporting the material, except originally in Tertiary time when they brought it down to make the coastal plain sediments.

* This adopts Ransome's term used for submarginal ore converted to commercial ore by secondary enrichment.[9]

In turn, reasons for the insignificant deposits of heavy minerals of commercial importance on the North American coast, in spite of intense concentrating conditions in the ocean itself, reflect this exceedingly low tenor of heavy minerals in the sands of the coastal plain sediments, which have been exposed to marine erosion, and in material brought down by the rivers.

The two deposits in Florida now being worked are not on or near modern beaches. One of these lies between Jacksonville and the coast, and the other 50 miles inland, at Trail Ridge on the former site of Camp Blanding. Both are large, but they consist of very low-grade sands. In age they are probably older than the Brazilian. The writer believes that they resulted from some special conditions of sedimentation which, because of the antiquity of the deposits, are difficult to interpret in detail. The Jacksonville deposit is related to an abandoned channel of the St. John's River and may have formed at a time when the mouth of that river was in that vicinity. The height-of-land called Trail Ridge was, in early Pleistocene time, the entire existing peninsula of Florida. It was a sand spit directed toward that hill now called Iron Hill near Lake Wales, on which the Bok Tower stands. Iron Hill was at that time an island in the sea (ref. 10, fig. 12, p. 35). The deposits of heavy minerals formed on the west or the gulf side of this spit.

The small beach deposits formerly worked at Ponte Vedra, near Jacksonville, and recently at Vero Beach, Fla., are storm-line deposits (ref. 11, p. 1592).

Black beaches on many volcanic islands, like those in the West Indies, formed exactly as case 3, by the undermining of the cliffs by waves. The source material (lavas), however, contained no ilmenite of acceptable commercial quality but did yield an excess of other black minerals of no value, such as titaniferous magnetite, augite, etc. The black beaches and dunes of New Zealand resulted from similar conditions of fluvial erosion and marine sorting as in Brazil, with the streams and waves working on a source material of volcanic ash very rich in titaniferous magnetite. In New Zealand where ancient and active dunes are of very large size, wind played and continues to play a very important role. Vegetation on the Taranaki coast of New Zealand where the deposits occur must have been almost nonexistent after a violent volcanic eruption of Mt. Egmont. The wind is still too strong along that coast to permit vegetation to get a start on the dunes in many places, as at Nukumaru, for example.

The layers of rutile-zircon sands on the eastern Australian coast described by Fisher[12] are elevated beach deposits, derived from the erosion of old sediments. There was little ilmenite in the source material.

Description of Individual Deposits in Brazil

The deposits in Brazil can be grouped geographically, as follows:

1. Those south of Vitoria in the States of Rio de Janeiro and Espirito Santo, which are not only the most numerous but include the largest individual ones.
2. Those in Espirito Santo, north of Vitoria.
3. Those in Baía, principally at Guaratiba and Comoxatiba. These deposits are of intermediate size and contain ilmenite of a quality comparable to the Indian.
4. Those along the hump of Brazil in the States of Paraíba do Norte and Natal.

Because of the number of individual deposits, it is not possible in any paper of moderate length to describe each individually. Such descriptions must remain for a government monograph. Since detailed maps and drilling records have been filed with the Brazilian government, the information is available to interested persons.

Deposits South of Vitoria: The deposits south of Vitoria can be divided into those south of Barra do Itapemirim and those north, mainly in the stretch of coast between Anchieta and Guarapary.

Those south of Barra do Itapemirim are: the single deposit of moderate size at Bôa Vista de Siri, and the very large deposits, of which twenty-two were mapped, lying between Barra do Itabapoana* and the mouth of the Paraíba do Sul at São João de Barra. With one or two exceptions, these numerous deposits are all of the elevated bar type.

* This name, obviously Indian in origin, is really very easy to pronounce—Ita-ba-po-ana.

The Itabapoana River is of very moderate size and forms the boundary between the States of Rio de Janeiro and Espirito Santo. The town of Barra do Itabapoana is a small port from which sugar and mandioca are shipped in very small coastal vessels. Offshore anchorage is unsafe. The town is reached by a road which is passable except in very wet weather from a station on the Leopoldina Railroad called Travessão, which is about 16 km north of Campos.

The deposits begin at the very edge of the town of Barra do Itabapoana, and extend south along the coast for over 20 km. They were designated by

letters, which run from A to V. Most of them are narrow, long ridges with a moderate surface expression. Although a little black sand can be seen in the surface sand, the rich layers in most cases are buried under 50 cm or so of barren or low-grade sand. Some of the deposits are small and others only a few hundred meters long. Two others, called P and U, are each nearly 2½ km long. The width varies, but the high-grade zones are not over 20 to 50 m wide. The depth to barren sand is less than 4 m in most places. The grade of the sand varies through wide limits, but for all deposits in the area the calculated average is 32 pct heavy mineral, which includes all minerals over 2.9 in gravity. Of this heavy mineral assembly about 55 pct is ilmenite; 5 pct rutile; 25 pct zircon; and 5 pct or more, monazite.* The balance is composed of other heavy minerals, principally tourmaline, staurolite, sillimanite, kyanite, and corundum. Spinel is abundant in the Itabapoana sand, but rare or absent in that of all of the other deposits.

* Exact quantities of monazite are impossible for the writer to give because of sampling difficulties. In splitting down a large composite to obtain a small sample for analysis, the monazite is not easily controlled since it is heavier and most of the grains are so much finer than those of the other minerals. The writer was sure of his results only in cases where a large sample was concentrated by ore-dressing methods, and all of the products weighed and analyzed, preferably with a Geiger counter.

Because of the attenuated nature of the deposits, with a low tonnage per meter of length, with no usable roads in existence parallel to the beach, with no safe anchorage nearer than 50 km, and two wide rivers to cross within this distance, and no developed power, the Itabapoana deposits present problems for mining and shipping in spite of the extent of the reserves.

The deposit at Bôa Vista de Siri is an interesting example of a single, isolated bar-type deposit which lies close to the barreira. Fig. 5 is shown to illustrate this only as a typical occurrence. Bôa Vista can be reached via the beach from Marataíses, a resort town a short distance south of Barra do Itapemirim. The beach can be traveled in a jeep very readily at low tide. It is one of the few sections of beach along the Brazilian coast visited by the writer that can be negotiated in a wheeled vehicle.

This deposit was worked for monazite by an Alsatian named Spitz, around 1910 to 1912. How the monazite was shipped in those days is not known. Some parts of the deposit are quite rich in monazite, but the total tonnage is not large. The deposit has not been worked in recent years. It belongs to "Mibra."

The deposits between Barra do Itapemirim and Vitoria are more noteworthy for their number than for their size, but all of the recent production has come from this group. From south to north, the deposits south of Anchieta are known as Caju and Piuma, and south of Guarapary they are Paraty, Ubu, Maíba, Ubahy, Meaípe, and Lima. North of Guarapary is a deposit, presumably exhausted, which supplied the bulk of the tonnage produced to date by "Mibra." Several kilometers further north is one at Ponta da Fruta that is quite different mineralogically from all the others in southern Brazil. In this deposit the "ilmenite" is similar to a true ilmenite, since it analyzes about 54 pct TiO_2, and contains principally ferrous rather than ferric iron, thus differing from arizonite which is the titanium mineral of most of the deposits. Here "Fomil" began production in 1946, principally for zircon.

Fig. 6 (above)—Dalas, or rockers, used in concentration of monazite.

Although primitive, they effectively produce a concentrate acceptable to the trade, with a moderate recovery. This plant is at Paraty.

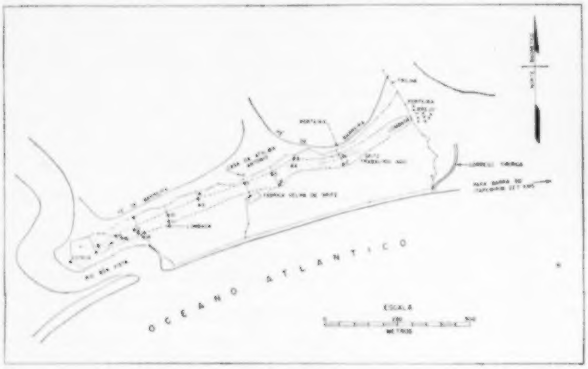

Fig. 5 (left)—Outline map of Bôa Vista de Siri, State of Espirito Santo.

Another peculiarity of the deposits in the vicinity of Guarapary is a rather high garnet content. Garnet is a rare accessory in the other deposits. The deposit at Ponta da Fruta is unusual also in that it contains considerable andalusite.

All of the deposits in this section are of the elevated bar type except the one at Maíba, which is a modern beach placer. Some of these deposits, though small, are rather rich in monazite, and were worked for that mineral alone in a period around 1910 to 1914 (by the Germans), and more recently by a local company, and later by Lindsay Light and Chemical Co. of Chicago. Some thin layers can be found that are so rich in monazite that they are very yellow in color. Since monazite is so much heavier than the other minerals, a concentrate of it can be made (with low recovery) by gravity methods alone. The earlier operators used a hand-operated device called in Portuguese a "dala" (fig. 6). A few shovels of sand are placed in the dala, in which a strong current of water is flowing. The dala is rocked like a baby's cradle, which helps to shake the heavy minerals to the bottom, and the light ones to the top, and the current of water causes the light grains to migrate more rapidly to the lower end.

"Mibra" has a plant at Guarapary consisting of tables and magnetic separators. "Fomil" installed tables and electrostatic machines as well as magnetic separators.

"Mibra" and "Fomil" have shipped the bulk of

their production from Vitoria, having transported it there partly by launch, and partly by truck.

Deposits North of Vitoria: Several deposits occur north of Vitoria, but south of Ara Cruz, and can be reached by road. These are known as Carapebús, Capuba, Jacareípe, and Bôa Vista de Nova Almeida. Carapebús was worked at one time during the 1910 to 1912 era, and is attractively located because shipments could be made from the point called Ponta Piraem, at the extreme south end of the deposit, by barge across the relatively protected waters of the outer harbor of Vitoria. Capuba and Jacareípe are contiguous deposits of the long attenuated bar type. Bôa Vista is of insignificant size.

The deposits north of the river at Ara Cruz (Rio Piraqué) are inaccessible except by trail, but one called Saí is of interest because it is fairly large and is the best example of the elevated beach type. The black sand lies at the base of the barreira, in a thin layer, sloping out toward the sea. It represents an old marine placer exactly like the modern ones at Maíba and Comoxatiba, but now, due to the 5-m uplift, the "beach" is no longer washed by the waves.

The deposits in the area are uninteresting since the ilmenite is mixed with magnetite altered almost completely to hematite, but still weakly magnetic, and the best grade of concentrate that could be made analyzes only 53 pct TiO_2.

The last deposit of any importance in this section is one at Barra do Riacho about 8 km north of Saí.

Further north 25 km is the delta of the Rio Dôce, one of the big rivers of central Brazil. Its delta is spread out widely over the coastal plain, and in the delta are numerous layers of concentration of black minerals, dominantly titaniferous magnetite.

The mineral grains are much coarser than those in all of the other deposits, and garnet is the only other heavy mineral of importance. The best ilmenite concentrate made analyzed only 37 pct TiO_2.

The deposit is intriguing because of its size and geologically interesting because the mineral association shows how essential the period of deep weathering of the old land surface was to the later formation of a high-grade deposit. The weathering, by chemical decay, eliminated the magnetite and other objectionable minerals, thus enriching the soil in ilmenite, zircon, rutile, and so on.

About 20 km up the Rio Dôce is a group of lakes (fig. 2) that have a great deal of geological and historical interest, and some deposits of insignificant size which, however, contain ilmenite of the highest quality in the southern field. These lakes are lagoons or land-locked estuaries, left far inland by the outward rapid construction of the delta of the Rio Dôce. Their eastern shores are actually elevated sand bars built across the mouths of small streams, when that was the ocean shore. The area has historical interest in that, following the American Civil War, a colony of Southern white families sought refuge from Northern "carpetbaggers" by emigrating to Brazil. A number settled around these beautiful lakes, but the pioneer economy of the day was too harsh and the settlement failed. Today there is no apparent record of this colony, neither in the buildings nor in the memory nor cultural inheritance of the people. A little earlier, the great emperor, Dom Pedro II, had come to Lagôa Juparana several times for holiday, and an island in the lake is known as "Ilha do Emperor." The logistics of transporting the imperial entourage to the region in the 1850's must have been a problem, and royalty must have endured physical hardships to which few vacationists today would submit. The writer spent a night in a bayou of the delta trapped in a dug-out "canoa" by "Agua Pé" (water hyacinths), and his bout with the mosquitoes, in spite of nets and repellants, was unpleasant. Nights must have been equally difficult a century ago, and in those days there were no outboard motors to propel dug-out canoes at dizzy speeds, as today! The black sands of Lagôa Juparana lie on beaches at the foot of the barreira which lines the shore of the lake, and in a few sand bars across bays. The tonnage, unfortunately, is too low for economic consideration.

Deposits in Baía: The short zone of favorable areas in southern Baía is so far from the other areas, north and south of Vitoria, that an operation to be established in Baía would be so isolated from the others that it would be completely separate.

Furthermore, the two known deposits are too far apart (about 40 km) to be operated together without some complications. Since the reserves of each are large enough to sustain only a moderate size operation, production costs will be increased by small volume. The northern one of the two deposits, called Comoxatiba, is a modern marine placer, and at high tide the waves dash against the base of an unscalable cliff, thus restricting the daily hours of potential working and limiting the kind of equipment to be used to the most mobile of units.

Other factors that will delay the development of these Baían deposits are the shallow depth of the channel to Caravellas, which limits access to boats of 12-ft draft, the lack of rail connection to Rio, and the high incidence of malaria in the district. All equipment used in construction of the huge military airport at Caravellas was brought from Rio by barge and unloaded at a beach head established below Caravellas.

The coast north from Caravellas is marked by almost continuous reefs composed of layers of the Tertiary sediments which are cemented by iron oxide. Although these reefs present hazards to navigation and have caused many disastrous wrecks, nevertheless they provide protected waters alongshore for barge loading from the beaches. Protected anchorages exist offshore for ships to anchor while loading from barge to steamer. Thus shipping of ilmenite and the associated minerals can be accomplished off the beaches, just as it is done in India.

The southern deposit is near a little point called Guaratiba, and a few kilometers north of the attractive little fishing village of Alcobaça. The deposit is an elevated bar, 5600 m long, and 25 to 200 m wide. It is the longest bar with continual sand of workable grade that was explored on the Brazilian coast. The deposit is covered with a forest of large trees. This is unique since all of the others lie in fairly open grassy land, or in cultivated fields. Another unusual feature of the Guaratiba sand is the presence in it of andalusite as the most abundant accessory mineral. The quantity of monazite present is only moderate. The ilmenite analyzes better than 61 pct TiO_2, and hence is of better quality than the Indian. An analysis of a big sample shows the ilmenite actually to be nearly pure arizonite with only 2.5 pct FeO. Had the tonnage of reserve in the Guaratiba deposit been two or three times larger, development of this deposit could be most attractive.

The Comoxatiba deposit lies 40 km north, near a village of that name. Until recently the road north from Alcobaça was practically impassible, although the writer made it once in a Model A Ford, with the help of several men with axes. Near Comoxatiba the sea had eroded all of the elevated strand below the old sea cliff, and is gnawing away anew at the Tertiary beds. Being marked by freshly exposed surfaces, the cliffs colored by the reds and buffs of the argillaceous sands, are conspicuous far out to sea.

Like all deposits of the beach type, the black sand belt is confined to a cove, the concavity of which has prevented lateral migration of the heavy minerals. The deposit is over 3 km long, and consists of the top layer of the beach, 1 to 2 m thick, which in many places is so rich that it is very black, with thin yellow streaks of monazite. At low tide the beach is 15 to 30 m wide. The sand is richer in monazite than any other of the larger deposits. The ilmenite is probably not as high grade as that of the Guaratiba deposits, although the writer has only one analysis available, made on the most magnetic fraction. This analyzed only 57.14 pct TiO_2.

This deposit was worked about 1903 by John Gordon, who shipped the monazite in ballast to Germany. He concentrated the mineral in sluices. It is believed that he shipped the considerable quantity of about 25,000 tons.

An interesting fact about the Comoxatiba deposit is that it continually renews itself by the progressive destruction of the cliff of Tertiary sands. How rapid is the rate of renewal is not established, but it is possible that much of the monazite now on the beaches is a new accumulation since Mr. Gordon's day.

Deposits in Northeast Brazil: Search for deposits along the "hump" of Brazil followed a hunch developed after discussing possible potential areas with Dr. Glycon de Paiva of the Brazilian Geological Survey. A trip was made to Natal by car from Recife. This section of the Brazilian coast has a better developed transportation system than further south, since the terrain is not so difficult.

The first indication of occurrence of ilmenite in the red sands of the Cretaceous beds was in the river shore near Cabadello, the port of São João de Passôa in the State of Paraiba do Norte. These beds are probably too low grade to work. At Natal the same red beds occur and black mineral is seen in the ditches within the town itself, but no important accumulation was found along the coast. The captain of the Port of Natal, who had lived at the fishing village of Cunhaú in the southern part of the state, recommended a look there. This little port is 50 km south of Natal, to which it is connected by a passable road through Canguaretama. The channel into the port is too shallow for boats larger than 100 tons.

The black sands were found to lie along the channel on both sides, and on the beaches north and south of the village for several kilometers. There is a big dune deposit between the sea and the old sea cliff. The tonnage in this dune is very large, although the average grade of the sand is low.

Ilmenite and zircon make up 85 pct of the heavy mineral content. Monazite and rutile are present in insignificant quantities.

The ilmenite analyzes only 56.2 pct TiO_2, and carries 17.8 pct FeO and 21.6 pct Fe_2O_3.

Description of the Minerals

The minerals of the Brazilian beaches are as follows:

Mineral	Pct of Heavy Mineral
Magnetite	Trace to 3
Ilmenite	35 to 75
Monazite	1 to 20
Zircon	5 to 35
Rutile	½ to 5
Sillimanite	1½ to 5
Kyanite	½ to 2
Corundum	Trace to ½
Spinel (several varieties)	¼ to 5
Staurolite	Trace to 3
Others	Trace to 3

The "others" include garnet, tourmaline, andalusite, hornblende, etc. Garnet is abundant in the beaches around Guarapary, and andalusite in the sand at Guaratiba. Spinel is abundant only at Barra do Itabapoana. The accessory silicates in all Brazilian beaches are much less abundant than they are in the Florida sands, for example, in which staurolite and tourmaline make over 25 pct of the "heavy mineral" (ref. 11, p. 1593).

The ilmenite, except at Ponta da Fruta, and probably part of that in the Natal deposits, is of the arizonite variety, having the theoretical formula Fe_2O_3 3 TiO_2. In another article the writer (ref. 6, p. 1043) has discussed the name arizonite as applied to the beach mineral found in India, Brazil, Florida, and other places. The Brazilian mineral analyzes from 2.5 to 6 pct or more of FeO. The iron content runs from 30 to 34 pct.

Comparative analyses of various "arizonites" are as follows:*

	TiO_2	Fe	FeO	Fe_2O_3	Cr_2O_3	V_2O_5
Indian (Quilon)	60.4	24.7	9.55	24.64	0.17	0.36
Floridian (Trail Ridge)	64.8	21.9	3.4	27.5	N.D.	N.D.
Brazilian—500 ton shipment from Guarapary in 1942	53.9	25.12	6.68	28.5	0.05	0.45
Itabapoana—average sample	57.0	27.0	3.60	34.8	0.08	0.22
Carapebus	58.06	25.8	5.92	30.46	0.03	0.44
Guaratiba	61.4	24.3	2.5	31.90	0.07	0.15

* All of these analyses were made in the analytical laboratory of the pigment plant at Baltimore, Md., of E. I. du Pont de Nemours and Co.

The "penalty in use" to pigment manufacturers because of the lower TiO_2 content and the higher ferric iron of Brazilian ilmenite is serious, and reduces its competitive value appreciably.

A factor contributing to this lesser attractiveness of Brazilian ilmenite is the absence of altered grains, higher in TiO_2, which boost the average quality in both the Indian and Floridian sands. The brown opaque grains which are weakly or almost nonmagnetic in Indian and Floridian sands assay 75 to 80 pct TiO_2, whereas in Brazil most of the brown grains are nothing but limonite, and only boost the objectionable content of ferric iron. The altered grains in Florida are described by Creitz and McVay.[12]

The Brazilian zircon is badly stained to a brownish color with organic matter, but this coating calcines off readily. Analyses showing 64 and 65 pct ZrO_2 were made on laboratory concentrates indicating that the material is of acceptable quality. The usual trouble was experienced in reducing the TiO_2 analysis of the zircon concentrate to acceptable limits, but this problem plagues producers from all sources and the Brazilian is no more difficult to clean than others.

The rutile is a good quality, and analyses of concentrates indicate that a grade of 95 pct TiO_2 could be attained regularly.

The monazite is low in ThO_2 as compared with Indian but carries as much of the other oxides. Analyses are as follows:

Source	Oxides of the Cerium Group	ThO_2	P_2O_5	CeO_2
Barra do Itabapoana	64.43	5.01	17.85	28.67
Barra do Itabapoana	64.50	5.23	15.57	28.81
Sai		6.89		30.04
[a]Travancore, India[c]	60.36	10.22	26.82	31.90
[b]Travancore, India[d]	61.73	8.65	26.50	Not Reported

[a] and [b] Analyses. [a] and [b] are reported by Houk,[14] table 4, p. 6 of the reference.
[c] Described as "sand from Travancore, India".
[d] Isolated from a concentrate from Travancore, India. No analyses as the content of cerium is reported as Ce_2O_3.

No concentrates were made of the corundum, nor of the aluminum silicates, although this problem should not be difficult since other gangue minerals of medium specific gravity and low magnetic permeability are present only in moderate amounts.

In grain size, the Brazilian sand is fairly coarse, as compared with Indian, and particularly with Floridian. Analyses shown in table I are typical.

Table I. Screen Analyses[a]

Ilmenite

Mesh	Barra do Itabapoana[b]						Guaratiba	Cunhaú, Rio Grande do Norte
	S-1	V-2	E-3	T-2	P-9	B-7-12	B-6	
+48	11.92	12.89	55.78	10.53	8.64		32.1	
+60	6.30	8.71	1.22	7.93	8.91	4.6	22.4	18.73
+80	51.37	41.30	29.20	57.57	49.20	59.6	32.7	36.08
+100	10.87	17.36	9.65	10.63	14.87	19.6	9.6	29.04
+150	16.17	17.06	8.58	11.01	15.60	13.9	3.2	15.09
—150	3.35	2.69	1.37	2.27	3.00	2.3		0.99

Mesh	Zircon			Monazite			
	Barra do Itabapoana		Guaratiba	Mibra Commercial Shipment[c]	Barra do Itabapoana		Guaratiba
	B-2-29	B-7-14	B-6		B-7-13	B-2-26	B-6-11
+60		2.37	10.10	46.09	0.39		0.59
+80	24.0	25.72	11.47	37.59	16.14	15.07	2.76
+100	33.5	25.22	34.87	13.19	24.02	34.40	21.65
+150	38.2	40.70	37.50	2.85	42.91	46.88	70.07
—150	4.1	6.24	6.11	0.27	16.54	3.64	4.92

[a] Tyler standard sieves were used.
[b] Letter designates the particular deposit, number is the individual composite made.
[c] This analysis implies the concentration process used fails to recover the fine monazite.

Concentration Problems

A number of "semi-works" tests on concentration of the Brazilian sand were conducted. Flowsheets following the Florida practice (ref. 6, p. 1063) are practical, in spite of the high tenor of the crude ore in heavy minerals. Preliminary gravity concentration in spirals produces a high-grade heavy mineral concentrate with over 90 pct recovery of these minerals. Electrostatic separation on machines of the Carpenter type separate conductors (ilmenite and rutile) from nonconductors, which are all the rest of the minerals. Ilmenite is separated from rutile on magnets of the induced roll type. Zircon is concentrated readily by respiraling the nonconductors, followed by final cleaning of the semiconcentrate on electrostatic machines and on magnets.

The monazite can be recovered by a combination of gravity and magnetic methods, as in Travancore, and some encouraging but very preliminary tests were conducted on flotation of the monazite, for which credit should be assigned to J. C. Detweiler, Chief Chemist of the Humphreys Gold Corp., which operates the Florida deposits.

The only sand that gave trouble in concentrating was that from Saí, Barra do Riacho, and the delta of the Rio Dôce. The difficulty with these sands is because the ilmenite is mixed with much altered magnetite which has about the same magnetic permeability as the ilmenite.

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Kaolin Production and Treatment in the South

by Paul M. Tyler

YEAR after year, the kaolin industry of the United States has been setting new production records and making better products. High-grade paper, pottery, and rubber clays are produced in this country mostly in the South. Georgia alone contributes over 70 pct and South Carolina almost 20 pct of the total domestic output. Residual kaolin is mined in North Carolina, highly plastic but naturally sandy Tertiary (Eocene) potting clays are worked in north central

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Florida, and good white clays are produced in several other states, but the main sources of kaolin or china clay have been numerous deposits in the Tuscaloosa (Upper Cretaceous) formation. This formation of generally sandy sediments is called the Middendorf member in older geologic reports and corresponds in age with some of the New Jersey clays. As shown in fig. 1, it crops out almost continuously in a generally southwesterly direction across South Carolina and Georgia and into Alabama. Clay is mined from this formation in all three states but the principal producing centers lie within about 10 miles of a straight line drawn between Aiken, S. C., and a point about 10 miles south of Macon, Ga.

The white kaolins of the South were recognized and used prior to the Civil War but suitable treatment processes were not introduced until World War I when imports, chiefly from England, were curtailed. Although imports of high-grade clays were resumed after 1918, the domestic industry managed to treble its prewar production record during the early 1920's and has continued to grow. Whereas the 1909 to 1913 average total production in the United States was only 132,104 short tons valued at $705,352 f.o.b. mines, the output in 1948 was 1,-568,848 tons worth $19,756,738.

Paradoxically, it seems in retrospect that the early failure of the American industry to meet foreign competition was due to the superior quality of our sedimentary clays in their natural state. Kaolin, of course, is the principal decomposition product of feldspars which originate in acidic igneous rocks such as granite, aplite, alaskite, granodiorite, quartz porphyry, etc. English china clays occur in residual deposits and before they can be marketed they have to be treated to remove accompanying quartz, mica, and other impurities. Notwithstanding the relatively crude methods employed, the final product is a beneficiated clay which is subject to a certain amount of technical control as to quality and uniformity. Although the naturally concentrated deposits in Georgia and South Carolina contain some of the finest crude white kaolin in the world, even it can be made better by suitable treatment.

In recent years well over half of the high-grade china clay produced in the United States has been used in making paper. Some qualities of paper clays are still produced by the dry process, or air flotation, but the paper industry's specifications have grown so exacting that wet processing was adopted and more refined methods had to be perfected. Notwithstanding notable advances in clay-preparation technology during the past decade, or possibly because these advances have implemented and encouraged technologic changes in consuming industries, demand has grown for products of higher uniform quality than can be obtained from even the best natural deposits without rigidly controlled fractionation. Largely as a result of the wide adoption of machine coating for paper, the clay industry has been obliged not merely to eliminate virtually all mineral impurities but also to segregate the clay substance itself into narrow particle-size ranges.

By extraordinary coordination of sales effort and production technology, several Georgia companies manage to market a wide variety of specialized joint products but the commercial success of many producers depends upon their mining only the best parts of their deposits and then skimming the cream of this almost pure clay in order to obtain a maximum yield of kaolinite finer than about 2 microns in maximum particle size and possessing low viscosity as well as the more familiar attributes of suitable color and brightness, or reflectance.

To the casual visitor from another mineral industry, the kaolin mines and plants may appear to be

Fig. 1—Sketch map showing generalized outcrop of Tuscaloosa formation and other kaolin-bearing beds in southeastern U.S.

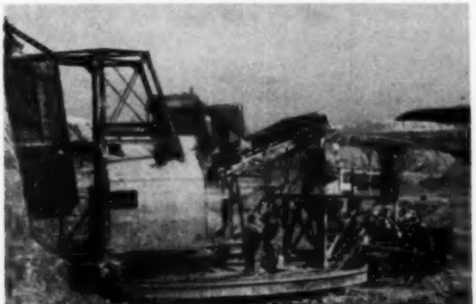
A general view of the kaolin industry of the country is given with special emphasis upon wet methods of beneficiation. Whereas dry milling procedures have become well-standardized, wet treatment technique differs widely and each company has its own flowsheet. Owing to the policy of secrecy in this highly competitive industry, plant descriptions are ruled out but methods and equipment employed by major producers are discussed for the principal treatment steps, crushing, blunging, grit removal, fractionating the clay, bleaching, filtering, and drying.

inefficiently operated and very wasteful of natural resources. However, it must be realized that the industry is in the throes of rapid development. Fundamental information is lacking on the nature and behavior of the several kaolin minerals. Consumers, too, often are confused as to specifications and cannot agree on suitable standards for making tests. Aided by new tools of modern research, some of the gaps in our knowledge are being filled, but certain of the difficulties in clay beneficiation are probably inherent. Many of them stem from the fact that colloidal or semicolloidal particles of the desired mineral have to be separated from other minerals and then sized to meet specifications that are not universally understood even in terms of performance in the consuming industries.

Apart from the emphasis on selectivity and avoidance of contamination with waste, kaolin mining does not differ greatly from other types of open-pit mining. Current practice in South Carolina and Georgia has been described in another paper.[1] Fig. 2 is a view in one of the large South Carolina pits. The only completely novel development is in Georgia. This is a continuous mining machine which digs the clay from the bank and delivers a dispersed slip or slurry to a pump and pipeline. This machine is

Fig. 2 (above)—Mixing clay before loading into trucks.
Barden Pit, J. M. Huber Corp., Graniteville, S. C.

Fig. 3 (below)—Continuous mining machine: crawler planer and mobile blunger.
Georgia Kaolin Co., Dry Branch, Ga.



really a combination of a shale planer and a blunger mounted on caterpillar treads and carefully coordinated. It is a great labor-saver but suffers from the disadvantage that it cannot be used in spotty deposits because it is less selective than a small power shovel or, as in earlier days, hand mining. Fig. 3 shows this machine ready to start a new cut.

Southern clays, as mined, are almost always rather acid. Before they can be completely dispersed the pH must be raised to around 6.5. Deflocculants are also added to stabilize the slip. Careful mining is necessary not only to avoid off color clay or excess grit but also to choose such portions of the clay in the bank as the drilling records indicate can be treated satisfactorily. Inclusion of even a small amount of a certain kind of clay, possibly bentonitic, will sometimes "poison" an entire batch. While methods may be worked out for correcting this, the best way to avoid trouble in the plant is to try to keep out the crudes that cause trouble.

Dry milling practice is becoming standardized. It is the only practice now employed in South Carolina where the principal product is rubber clay and is also used in Georgia for refractory kaolins, most potting clays, and to some extent for paper filler and other clays except paper-coating clays. The typical flowsheet is quite simple. Essential features are the large storage shed, the "Payloader" for reclaiming the crude clay from the concrete floor of the shed, the new clay shaver or "slicer" (described later) for shredding the moist clay, the oil-fired semidirect rotary drier (usually a Ruggles-Coles), two 5-roller, high-side Raymond mills with built-in whizzer attachments for fine grinding, and the Bates (St. Regis) valve bagger. For further details, the reader is referred to another recent paper.[2]

Clay Treatment

Virtually the entire North Carolina output of kaolin is used in pottery, mainly table ware and high-grade floor and wall tile. The geology of these primary deposits and recent mining and treatment practice has been summarized by Stuckey[3] and flowsheets and descriptions of several plants have been published by Hubbell.[4] Mica is a remunerative byproduct constituting 5 to 10 pct of the mine-run crude. It is recovered by tabling and froth flotation.

The Florida clay, intermediate in properties between a true kaolin and a ball clay, characteristically occurs below or only slightly above groundwater level. It is used entirely for pottery, electrical porcelain, and high-grade tile. The numerous known deposits, only two of which are currently worked, have recently been described.[5] The clay-bearing sands (containing less than 15 pct clay as mined) are stripped of overburden and then washed by hydraulic giants in front of a suction dredge which floats on the pond or artificial lake. The slurry, carrying about 12 pct solids, is neutralized and may be pumped to an Akins classifier floating on a barge in a separate pond as shown in fig. 4. Most of the associated white sand goes into this pond and can

Fig. 4—Akins classifier on barge in sand pond.
United Clay Mines Corp., Hawthorne, Fla.

be reclaimed. The classifier overflow or clay slip is further purified before reaching the filter presses. The filter cake is often quite brown but the dry clay is almost white and fires white.

Present-day Georgia practice for washing kaolin (fig. 5) does not appear to be developing a standard pattern. The exchange of ideas which characterizes technologists in other mineral industries is lacking. Published research deals with properties of products not with processes employed in obtaining the products. A legitimate criticism is that many clay operators used to try to solve their problems by cut-and-try methods. This condition is being remedied and ideas are being borrowed from other process industries. The steps in preparing high-grade kaolin for market are few in number. Although some features are unique most of the basic operations have a counterpart in other mineral engineering fields. The following discussion is general in order to avoid disclosure of conditions at individual plants. Only when the information is generally known by a company's competitors can the equipment or methods used in a particular plant be identified.

The obvious starting point for any wet treatment process, of course, is a water suspension of the clay. Since hydraulic mining is not practiced in Georgia, the first step is to crush to sizes that are small enough to break down readily in the type of blunging equipment installed. A blunger is any device that will disintegrate the clay and disperse it in water so that sand and other impurities can be dropped out. Mechanical purification may be followed by chemical treatment to eliminate iron staining and, in the case of paper-coating clays, the kaolin particles themselves must be segregated into suitable size fractions. Finally, the various fractions must be thickened, filtered, and (usually) dried.

Dry clay is easily crushed. Formerly, it was common practice to air-dry the material in a storage shed, but this involves a good deal of labor and is inconvenient as well as costly. Clay as mined ordinarily carries over 20 pct moisture and will clog any ordinary jaw, gyratory, or roll crusher. Many small to medium-sized plants use a specialized grinding pan known as a "slicer." The perforated bottom plate of this machine carries 32 to 80 knives, each in a separate slot like the blade of a carpenter's plane. As the plate revolves the action is similar in principle to that of cabbage cutters used in the home kitchen for making cole slaw. On rainy days, if the clay is unusually sticky, a little dry clay may be fed to absorb excess moisture. This machine handles up to 30 or more tons an hour, depending upon the diameter and number of knives, and reduces large lumps to less than 1-in. across. Figs. 6 and 7 show two types of these machines and fig. 8 shows raw clay being dumped into a machine from a payloader.

One of the largest companies stage-crushes lumps up to 6 ft in diam to a similar small size in two sets of Eagle crushers. This installation can handle 100 tons or more per hour. Powerful curved knives mounted on horizontal shafts chop the clay and feed it into the crushing zone. Similar machines are seen at brick-making plants. At other large plants, secondary crushing is done in the blungers. One company accomplishes the initial reduction with a combination of one corrugated and one smooth roll, another prefers a single-knobbed slugger roll opposed to a traveling breaker plate, still others use double rolls one of which is equipped with sharp cutting knives.

Blunger designs likewise vary greatly. As previously indicated several companies feed rather large lumps to the blunger which thus becomes a wet secondary crusher in addition to dispersing the clay in water with suitable electrolytes to form a creamy liquid or "slip." Horizontal blungers of the log-washer type may be used for this purpose; instead of paddles, however, they carry knives or curved beater arms. In another familiar type, beater arms are mounted on a rapidly revolving vertical shaft. The vortex-type blunger seems to be especially effective providing the raw clay has been crushed fine enough beforehand. This type may be compared to a wet cyclone in which the settlings at the bottom of the cone enter the suction of a centrifugal pump and are recirculated in a tangential stream in the upper part of the chamber but below the overflow. Small lumps of clay are thus disintegrated partly in the pump and partly by abrasion as they are pressed against the walls of the settling chamber by the whirling action of the slurry. The blunger used in the continuous mining machine employed by the Georgia Kaolin Co. at its Dry Branch mine is of this type. It delivers a smooth slip which is pumped to the sand-removal plant. Pebble-mill disintegrators would seem to have certain theoretical advantages but apparently are not

Fig. 5—Clay washing plant.
J. M. Huber Corp., Huber, Ga.

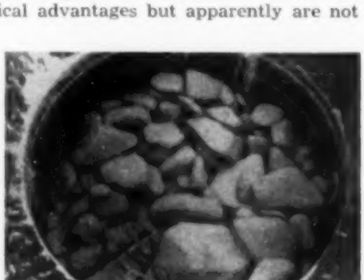


Fig. 6—View from top of 80-knife clay slicer.
Champion Paper and Fiber Co. plant, Sandersville, Ga.

used in Georgia plants, possibly because they might pulverize some grit.

Edgar Brothers Co. and the Georgia Kaolin Co. blunge their clay at the mines and deliver the slip, after removing grit, to their respective treatment plants several miles away through pipelines. The former company pumps material carrying 20 pct solids while the latter handles a 40 pct slip without difficulty. Other companies deliver raw clay to their plants in trucks or, in one case, by narrow-gauge steam train.

In accordance with English china clay practice, several companies conduct their slip through a series of sand boxes and mica troughs to remove grit. The density of the slip averages between 20 and 30 pct solids. Retention time in the system is about 30 min. In order to sluice out grit at intervals, usually once a shift, two or more series of troughs must be provided. The settlings are run to waste except at the extreme end where a separation of the coarsest clay fractions may be made. It is generally considered uneconomical to reblunge and resettle the clay to recover that entrapped in the sand settlings. Sand drags (flight conveyors) are used in two or more plants (fig. 9). Vibrating screens to remove tramp wood and to retain particles 200 mesh or over are installed at the discharge end of the drags. Several plants employ screens to remove the bulk of grit; hydroseparators are used for grit removal and also for fractionation. The material in the hydroseparators overflow may be fine enough for paper-coating clay, in which case the underflow is reblunged, resettled, and the second overflow containing coarser clay fractions is recovered for paper filler and other uses.

Two companies, The Georgia Kaolin Co. and Edgar Brothers Co., are licensed to use the Bird continuous centrifuge. The general design of this machine is well-known and certain of its applications have been sufficiently described.[8] Recent improvements have simplified construction, lengthened the period of operation between shutdowns for repairs, and facilitated controls without modifying basic principles. Its action is in harmony with Stokes' law except that the separatory force may be as much as 1000 times gravity. For certain uses, the clay passes through the Bird machine only once, thereby removing fine grit and possibly some of the coarsest particles or flocs of clay. To produce quality

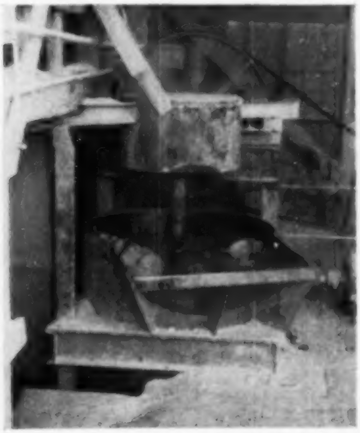


Fig. 7—Another type of clay slicer.

National Kaolin Products Co., Aiken, S. C.

Fig. 8—Dumping raw clay into slicer from payloader.

National Kaolin Products Co., Aiken, S. C.

paper-coating clays, however, the slurry is passed to a second centrifuge which rejects all particles over, say, 2 microns. Clean-cut separations may be made as low as 0.5 micron, if required.

The quantity of grit removed varies at different mines and in different parts of the same mine but the average in Georgia wet-treatment plants may be as much as 15 pct of the dry weight of raw clay treated. This compares with a rejection of about 1 pct at many South Carolina dry-treatment plants where only certain high-grade clays are mined and where there is probably some pulverization of quartz. In both states, the bulk of the grit is composed of extremely fine siliceous sand. Mica, however, may comprise as much as 30 pct of the total and minor amounts of other minerals may be identified. Recovery of the fine mica has been considered, appears practicable, but has not yet been accomplished commercially. Although presumably somewhat sericitic it possesses a fairly good sheen and general appearance. One reason for the retention of mica troughs at several plants, apart from their low initial capital cost, is the traditional argument that while they are prodigiously wasteful of labor and good clay, they do remove mica effectively. Mica flakes more nearly conform to Stokes' law in a slowly flowing stream than in a rising current type of classifier. Except perhaps in the Bird centrifuge, fine mica tends to contaminate certain clay fractions. One would suppose that it could be easily and profitably removed by froth flotation but certain operators insist that it is something of a problem.

Since few clays are white enough to be used for coating fine paper, even after being freed from grit, it is generally necessary to bleach them. This is a batch operation conducted in large wooden tanks. Processing data are carefully guarded but it is no secret that bleaching is ordinarily performed in a slightly acid (e.g. pH 4.6), moderately dilute pulp using a small amount of zinc hydrosulphite freshly prepared from sulphur dioxide and zinc dust. Alum has to be added to complete the reaction. The SO_2 is usually purchased from a single supplier who contributes technical advice. About 2 or 3 lb of the gas are used per ton of clay. Yellow or brown iron colorations (limonite?) yield readily to this treatment whereas pink or red stains (hematite?) are often difficult, if not impossible, to bleach out. Gray tones may sometimes be removed by suitable oxidizing agents but this type of treatment is seldom required on Georgia clays.

Conventional plate-and-frame filter presses, following thickening in concrete tanks, are standard at most clay-washing plants. Due to the high cost of labor and filter cloths, however, many believe that they are on the way out. Thickening offers no problems; if not already acid from the bleaching step, the pH is lowered by adding alum or sulphuric acid to cause flocculation. String filters are employed at one or more plants and several companies have been using continuous rotary vacuum filters of special design. Despite their high initial cost the latter seem to be gaining ground due to their extremely low labor requirements and large rated capacity. Although used principally on paper-filler clay, because this is somewhat coarser grained as well as a larger tonnage item, they are also said to be successful on the best grades of paper-coating clay. For use in the clay industry, however, these machines have to be built to order. Information as to optimum pulp densities, the best kind of weave for the filter cloth (preferably rayon), and types of cake-stripping mechanisms cannot be published.

Filter cakes contain about 30 pct moisture and may be dried in various ways. Proctor and Schwartz tunnel driers with perforated aprons may be employed either ahead of a rotary drier or alone to produce the finished product. Gas heat is preferred but is not generally available so steam pipes are used. The most common type of drying equipment is the indirect-fired rotary drier but a tray-type or a Sargent drier may be observed now and then.

For some uses clay is shipped "bone dry" (under 1 pct moisture); on the other hand some consumers accept material carrying 10 pct moisture. Paper-filler clay usually carries about 6 pct moisture and coating clay about 5 pct. Potting clays may be treated with measured amounts of Calgon or similar dispersant and sprayed into a Buffalo-vac drum drier, thereby relieving the pottery maker from the necessity of adding chemicals to prepare the slip.

In recent years some paper clay has been shipped in tank cars in the form of low-viscosity (e.g. 225 centipoises) slip containing 70 to 72 pct solids by weight. This product is made by agitating the filter cake with deflocculants; the resulting mobile liquid will stand for weeks without segregation. This mode of shipment simplifies operations at the paper mill and any additional cost for chemicals at the clay plant is offset by not having to put the material through a drier. Railroad freight increases, however, have penalized this practice so that it now seems destined for abandonment. The water content adds over one-third to the already high cost of shipping the clay from Georgia to mills in Wisconsin, Maine, or the Pacific Northwest.

A whole paper might well be written on the increased instrumentation and the development of more and more automatic controls in kaolin plants. Throughout all operations, the pressure on labor economy and products standardization have greatly expanded the number and variety of records to be kept. Every car of paper clay needs to be sampled and tested for color, acidity, moisture, grit, particle size distribution, and viscosity. But this is only the final check. Additional tests are made all along the production line, beginning before the clay in the bank is even stripped and continuing throughout the mining, blunging, and succeeding steps. A daily inventory of clay in process is taken by gauging tanks and pipelines. Owing to variable moisture content simple measurements of tonnage or volume

Fig. 9—Sand drags.

Georgia Kaolin Co., Dry Branch, Ga.

of raw clay crushed or blunged are not particularly accurate but once the clay is properly dispersed its progress is readily followed by first metering the water added and then registering the flow and specific gravity of the slip at strategic points.

Chemical analyses of raw Georgia kaolins show them to be highly pure. After washing and bleaching the principal impurity is about 1.75 pct of TiO_2. The question naturally arises whether the brightness of clay might not be further enhanced by removing this impurity. The measure of brightness in the paper-clay trade is the percent of light reflected at a certain spectral wave band. Tested in a color analyzer, the best clays show excellent reflectance at the red end of the spectrum but rather poor reflectance in the blue and green range. Bleaching, by eliminating much of the effect of iron, raises the reflectance materially and enough work has been done to show that removal of the titania would raise it even more. Under certain circumstances this can be accomplished by froth flotation but there may be other ways. On some crudes, however, even the laboratory method of electrodialysis does not work.

A certain amount of experimentation has been conducted on the froth flotation of kaolins. It is often possible to remove a considerable amount of off-color material from a batch of clay without making any significant improvement in the properties of the remaining clay. However, more experimentation with flotation is definitely indicated, using both anionic and cationic reagents. Electrophoresis, or electro-osmosis as hitherto practiced in Germany and elsewhere, has not resulted in any substantial purification traceable primarily to selective deposition or migration caused by the passage of the electric current. Electrophoresis (also called cataphoresis) usually amounts to simple dewatering; its use with the Bird centrifuge has been patented.

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Some Properties of Pseudowavellite from Florida

by W. L. Hill, W. H. Armiger, and S. D. Gooch

The physical properties, chemical behavior under thermal treatment, and fertilizer value of fluorine-containing pseudowavellite (hydrous calcium aluminum phosphate) that occurs as phosphate clay admixed with quartz sand in an extensive deposit near Bartow, Fla., were determined by experiments in laboratory, pilot-plant, and greenhouse. When the mineral was heated at 500°C, 80 pct of the phosphorus became citrate soluble. The heated product, however, deteriorates markedly under moist conditions such as obtain in the soil.

EXTENSIVE beds of a soft aluminiferous phosphate lie in the Bartow-Pembroke region of Florida. Although the extent of the deposit is unknown, it is thought that the equivalent of at least five million tons of material containing 25 pct P_2O_5 exists on one large mine property alone, and soft phosphates similar in nature are known to occur elsewhere in the state. This aluminum phosphate, unlike fluorapatite, is rendered fairly soluble in neutral ammonium citrate solution merely by heating it at a temperature below 600°C, a property made the basis of a process patent by the junior author.[1] The possibility of producing an available form of phosphorus from what has hitherto been a worthless natural phosphate prompted the Bureau of Plant Industry, Soils, and Agricultural Engineering to make a brief study of the Pembroke phosphate during the war years with a view towards determining the optimal temperature for heat treatment and the reactions involved in the process. The results of this study supplemented by data and information furnished by the junior author are presented in this paper.

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Description of Natural Phosphate: The phosphate with admixed quartz sand and siliceous matter occurs as a low-grade (8 to 9 pct P_2O_5) clay-like material that can be beneficiated cheaply by making a water separation of the very fine material from the sand, floating off the fine material, settling the suspension to a thick slurry, and finally dewatering the slurry by vacuum filtration followed by drying. The product dried at 100°C contains about 16 pct water and 25 pct P_2O_5, about 1/12 of which is citrate soluble. Partial analyses of representative samples of the crude ore and corresponding concentrate are given in table I.

The concentrate is light yellow in color and even after being dried at 100°C sticks very noticeably to laboratory grinding and screening equipment. Under the microscope the ground material is predominantly light brown crystalline aggregates with index of refraction near 1.61. The principal phosphate mineral constituent was identified as pseudowavellite with admixed phosphosiderite.[2] Pseudo-

Table I. Analyses of Natural Phosphate and Products Experimentally Derived from It in Pilot-Plant Operation

(Results are on basis of samples as received)

Sample No.	Type of Material	Total P_2O_5, Pct	Citrate-soluble P_2O_5, Pct of Total	CaO, Pct	Al_2O_3, Pct	Fe_2O_3, Pct	F, Pct	Moisture 105°C, Pct	Ignition[a] Loss 800°C, Pct
2334	Ore	8.67		5.40			0.73	0.40	4.05
2335	Concentrate	24.92	7.9	8.92	29.04	3.80	1.19	1.00	15.06
2336	Heat-treated concentrate	28.20	48.5	10.25	32.90	4.42	1.34	0.73	3.59

[a] Exclusive of moisture at 105°C. Result includes a small amount of organic matter and some volatilized fluorine, but only a trace of carbon dioxide.

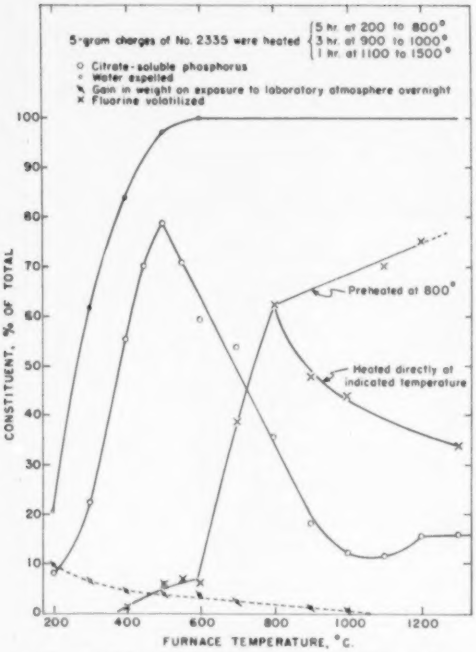

Fig. 1—Effect of ignition on composition and citrate solubility of concentrated phosphate.

wavellite is a hydrous calcium aluminum phosphate having the composition[5] $CaAl_3(PO_4)_2(OH)_5H_2O$* or $4CaO \cdot 6Al_2O_3 \cdot 4P_2O_5 \cdot 14\ H_2O$. If in the case of the concentrate (table I) iron and fluorine are disregarded and a figure for water is derived by correcting the total loss on ignition for volatilized fluorine, the composition corresponds with $3.35CaO \cdot 6Al_2O_3 \cdot 3.70P_2O_5 \cdot 18.4H_2O$. From this it is seen that in comparison with the formula derived from a structural analysis for pseudowavellite,[5] the sample is deficient in both calcium and to a less extent phosphoric oxide, but carries excess water.

* The formula given by Larson and Shannon[3] and by English[4] is $5CaO \cdot 6Al_2O_3 \cdot 4P_2O_5 \cdot 18H_2O$. Furthermore, pseudowavellite and deltaite, $Ca_2Al_2(PO_4)_2 \cdot (OH)_4 \cdot H_2O$, are isostructural and probably isomorphous,[5] both being uniaxial and positive with a rhombohedral unit cell. Indices of refraction of three specimens each are: pseudowavellite 1.618 and 1.623, 1.619 and 1.627, 1.622 and 1.631; deltaite 1.641 and 1.650, 1.630 and 1.640, 1.621 and 1.629.

Description of Heat-treated Concentrate: In the process for converting the concentrated phosphate into available phosphate,[1] the filter cake (about 55 pct solids) is heated at about 550°C in a rotary kiln. The resulting nodular product is cooled and ground to approximately 200-mesh. The ground material contains 27 to 28 pct P_2O_5. The citrate solubility varies with the temperature of treatment and with age and storage conditions, which are discussed later in this paper. A partial analysis of a typical sample is given in table I.

The heat-treated material has a somewhat deeper color than the concentrate dried at 100°C, is practically insoluble in water, and yields a suspension with pH 6.5 to 7.5. The freshly prepared product is an amorphous material with the exception of a scattering of stable minerals, mainly quartz. In nodular form it packs to a bulk density of 52 lb per cu ft, whereas the ground material packs to 42 lb per cu ft. The specific gravity of the powder, determined by the pycnometer method, is 2.56. The freshly prepared material carries practically no water and will remain workably dry after the addition of 20 to 25 pct water.

Laboratory Experiments: The concentrate in 5-g charges, No. 2335 (table I), ground to pass an 80-mesh sieve, contained in 50-cc platinum dishes, was heated at a series of temperatures in the range 200° to 1400°C in a well-ventilated muffle furnace for 1 hr or longer. The loss in weight on heating was noted, and the tendency of the heated material to gain weight from the atmosphere was determined by exposing the charge to the atmosphere overnight. The material was then put through an 80-mesh sieve and stored in a screw-cap bottle. Fluorine and citrate-insoluble phosphorus were determined on the heated products. Fluorine was determined by distilling an 0.5-g sample with perchloric acid, having due regard for the very slow rate of release of the last quarter of the fluorine from materials of this type, and titrating the distillate with thorium nitrate solution.[6] Citrate-insoluble phosphorus was determined with the use of 1-g samples in accordance with the official method for determining available phosphorus in fertilizers.[7] The citrate-insoluble residue was washed with 5 pct ammonium nitrate solution in order to minimize the tendency of the phosphate to run through the filter. Despite every precaution to keep the time and other factors constant, however, results of duplicate determinations often differed by as much as 0.8 pct. Contrary to the usual behavior of phosphates the "solubility" of the heated material in neutral ammonium citrate solution increases with the size of the sample (table II). A similar observation on aluminum phosphate from the Connetable Islands was made some years ago.[8]

Effects of Heat Treatment on Phosphate Concentrate: *Citrate-soluble Phosphorus:* The citrate-solubility of the phosphorus increased rapidly with the temperature at which the material was heated (fig. 1) up to 500°C, where the solubility reached nearly 80 pct of the total phosphorus, then fell off with further increase in the heating temperature to less than 20 pct at 900°C and remained between 10 and 20 pct in the temperature range 900° to 1400°C. Strong sintering of the charge began at 900°C. The material appears to be similar to the aluminum phosphate from Redonda Island studied by Morse[9] which showed a maximal solubility at 560°C. It is, however, unlike the calcium-free aluminum phosphate from Connetable Island,[8] which loses substantially all water of hydration at 105°C and attains a solubility around 60 pct that persists up to ignition temperatures as high as 800°C.

Expulsion of Water: The water was completely expelled at 600°C (fig. 1) and about 95 pct had been driven off at 500°C where the optimal solubility occurred. Up to this temperature the data

Table II. Effect of Size of Sample on the Result for Citrate-soluble Phosphorus

Sample No.	Type of Material	Total P_2O_5, Pct	Citrate-soluble P_2O_5 in Per Cent of Total Determined on Sample			
			0.5 g	1.0 g	1.5 g	2.0 g
2335	Concentrate	24.92	5.3	6.0	2.8	2.4
2336	Heat-treated concentrate	28.20	44.2	52.4	57.1	58.0

indicate a linear relationship between the citrate solubility of the phosphorus and the proportion of water expelled. Materials heated at the lower temperatures showed a marked tendency to gain weight on exposure to the atmosphere, a fact that is illustrated by the dotted curve in fig. 1. These figures were obtained by exposing the undisturbed charge; larger gains would probably be attained if the charge were stirred several times during exposure.

Fluorine Volatilization: Evolution of fluorine began at about 400°C (fig. 1), increased slowly with temperature to 7 pct of the total fluorine at 600°C, then very rapidly to about 63 pct at 800°C, where the material began to sinter, and either increased slowly or dropped off with further increase in temperature depending upon whether the charge was preheated at 800°C or heated directly at the indicated temperature. The small volatilization of fluorine below 600°C can hardly be a factor in the development of citrate-soluble phosphorus.

Reactions: Thermal-analysis curves (fig. 2) indicate that five reactions occur within the temperature range 200° to 900°C. Comparison of the heating curves with the dehydration data (fig. 1) shows that the first two reactions involve water loss. The first is presumed to be the loss of water of crystallization (around 200°), whereas the second involves the expulsion of hydroxyl water (around 400°). At a somewhat higher temperature fluorine, which replaces hydroxyl water, is rapidly expelled. This is doubtless the third reaction. The exothermic reactions (IV and V) are presumed to correspond to a crystallization of the amorphous material formed at lower temperatures. The latter view is supported by X ray powder diffraction photographs of the heated materials, which show that the material becomes amorphous at 400°C and that crystallization begins at about 600°C. One of the compounds formed in this crystallization is apatite, which was present in material heated at 800°C and had become a prominent phase in material heated at 1000°C. Finally, it will be noted that the maximal citrate solubility of the phosphorus occurs in the temperature range where the material is almost completely amorphous.

Deterioration of Heated Material: The heat-treated concentrate can be cooled by quenching the hot furnace charge in water without affecting the solubility of the promptly dried (100°C) product. Nevertheless, the citrate solubility undergoes a gradual decline with time. For example, a batch prepared in August and stored in paper bags contained better than 20 pct of citrate-soluble P_2O_5, which had declined to 18 pct by October, and to 15.5 pct by the following March. A similar, though slower, reversion has been observed in heat-treated aluminum phosphate from Redonda Island.[9] In the present case deterioration was observed even in material that was stored in a bottle and sealed to prevent the ingress of moisture. In view of the fact that the increased solubility occasioned by heat treatment is due primarily to the conversion of a crystalline material to the amorphous condition, the loss in solubility in the absence of water might be attributed to a spontaneous crystallization of compounds that are less soluble than the amorphous parent material. This possibility was not investigated.

Although directly comparable results are not at hand, evidence points to an accelerated decline in solubility when the material is kept wet with water. Thus, two laboratory preparations obtained by heating a concentrate at 450° and 600°C, respectively, that contained around 20 pct citrate-soluble P_2O_5

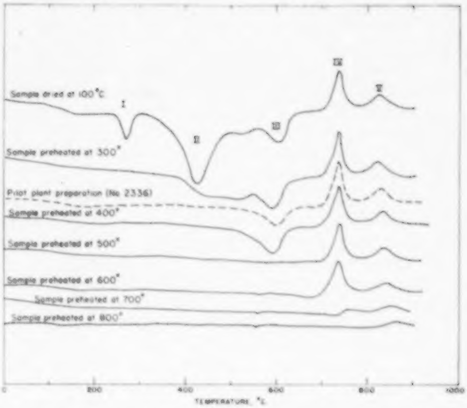

Fig. 2—Heating curves of concentrated phosphate (No. 2335).
Rate of heating was 12°C per minute.

Table III. Yields of Millet in Greenhouse Experiments with Heat-treated Concentrate and Double Superphosphate

Applied Phosphate — Type	P_2O_5, Pct — Total	P_2O_5, Pct — Solubility[b]	Average Dry Weight of Plants[a] in Grams per Pot of Total P_2O_5 Applications of — 50 lb Per Acre	100 lb Per Acre	200 lb Per Acre	400 lb Per Acre
Experiment I on Evesboro sandy loam[c]						
Concentrate No. 2335	24.9	7.9			1.1[d]	
Heat-treated concentrate No. 2336	28.2	48.5	4.0	8.6	11.2[d]	
Double superphosphate	48.7	96.0	13.1	19.3	19.7[d]	
Experiment II on Evesboro sandy loam soil[e]						
Double superphosphate	48.7	96.0	3.11	3.69	5.63	6.25
Heat-treated concentrate No. 2476	28.9	79.6	1.11	1.77	2.51	3.22
Same reverted by wetting (No. 2477)	28.9	63.7	0.98	1.58		

[a] Differences in average dry weights required for significance are:
Experiment I—1.75 at 5 pct level (19:1 ratio) and 2.65 at 1 pct level (99:1 ratio).
Experiment II—for 50 and 100 lb rates, 0.80 at 5 pct level and 1.05 at 1 pct level; for 50 to 400 lb rates, 0.85 at 5 pct level and 1.12 at 1 pct level.
[b] Per cent of total P_2O_5; neutral ammonium citrate method.
[c] Average dry weight of plants without applied phosphorus was 1.3.
[d] Rate of application was 150 lb per acre.
[e] Average dry weight of plants without applied phosphorus was 0.44.

(fig. 1), were subsequently moistened, kept wet for 48 hr and then dried at 105°C. This treatment increased the water content (+105°C) of the samples by 3.1 and 1.8 pct, and lowered the soluble phosphorus by about one fifth of the total phosphorus. Accordingly, it is obvious that these wet samples deteriorated within 48 hr to approximately the same extent as was observed over a period of seven months in the dry material cited above.

Fertilizer Value of Heated Concentrate: The fertilizer value of a concentrate, two heat-treated concentrates, and a reverted material, in comparison with double superphosphate, was determined in greenhouse experiments, in which millet was grown on soil that responded to phosphate applications. The greenhouse procedure is described elsewhere.[10] In experiment I, planted February 26 and harvested April 9, 1945, millet was grown on Evesboro sandy loam soil (pH 5.0) from the Beltsville Research Center, Beltsville, Md. In experiment II, planted March 7 and harvested April 11, 1949, millet was grown on Evesboro sandy loam soil (pH 4.8) from Beltsville, Md. In both experiments the soil was limed to give a pH of about 6. Four replicates were provided for each treatment. The average yields are given in table III.

The heat-treated material, though greatly superior to the raw concentrate, was in every case inferior to double superphosphate and the differences between yield responses are highly significant. The responses to the heat-treated material in comparison with those to superphosphate are somewhat lower than would be expected on the basis of the citrate solubilities. The response ratios are around one fourth to one third in comparison with a solubility ratio of about one half, a situation that could well be attributed to deterioration of the heat-treated material as it lies in the moist soil. Experiment II provides a comparison of a heat-treated product with a portion of the same that had been reverted by keeping it wet for 48 hr in the laboratory. Although the yield responses to the reverted material are in both instances lower, the differences are not statistically significant. Apparently deterioration occasioned by the long contact of the heat-treated material with moist soil all but obliterated the effect of any initial difference in availability of the phosphorus to the growing plants.

Acknowledgment

The authors are indebted to S. B. Hendricks and J. G. Cady, of the Bureau of Plant Industry, Soils, and Agricultural Engineering, U. S. Department of Agriculture, for the optical examination of the laboratory preparations, and to R. A. Nelson and Cecil Pinkerton, formerly of the Bureau, for the differential thermal analyses and chemical analyses, respectively.

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Correction

In the January 1950 issue: TP 2629 H. *Chromite and Other Mineral Occurrences in the Taştepe District of Eskişehir, Turkey* by Ferid Kromer, p. 108, under the subheading "Mineral Occurrences: Chromite," the last sentence of the first paragraph should read, "However, a new lode, very recently plotted, in the Taştepe mine averages 50 pct Cr_2O_3, 4.6 pct SiO_2, and 14 pct FeO." The author in making the correction added the following information: The production of about 4000 tons which came from this new lode has given the following analysis: 48.19 pct Cr_2O_3, 13.36 pct FeO, and 7.82 pct SiO_2. All the production went to Austria, on the basis of the above analysis.

An Electronic Tramp Iron Detector for Ore Conveyor Belts

(With Discussion)

by C. M. Marquardt

Tramp iron and steel moving on a conveyor belt cause small currents to be generated in a coil situated in a strong magnetic field, which are converted to an alternating current and are amplified. The output voltage from the amplifier fires a thyratron with a relay in its anode circuit, which actuates a howler and simultaneously drops a spot of marker material on the belt.

THE problem of tramp iron removal from moving ore belts is a long standing one. When heavy ore streams are carried on a belt, magnetic pulleys and strong surface magnets fail to remove tramp buried in the ore stream. Tramp iron buried in the ore stream cannot be pulled from the bed by surface magnets and is carried past the magnetic pulley by the falling ore stream.

With the increased use of detachable bits the problem of the detection and removal of tramp has become more necessary and difficult.

Several types of tramp detectors have been developed. It would serve no practical purpose here to review exhaustively the literature on the subject. Methods of tramp detection used are: (1) The magnetic method, wherein the small current generated by the magnetized tramp passing a coil is used. (2) The unbalanced oscillator method wherein the tank circuit of a stable radio- or audio-frequency oscillator is unbalanced causing a change of frequency or a change in plate current due to changes in eddy currents, hysteresis, or dielectric because of the presence of tramp. (3) The bridge methods wherein the impedance of one leg of an alternating current inductive bridge is changed due to the presence of conducting tramp.

Each of these methods of tramp detection has its field of usefulness. The second and third methods are excellent for oxide iron ores, coal, sand and gravel, grain, etc. However, these latter methods are not suitable for use on ores that contain rich conducting sulphides, since a large piece of pyrite or galena will cause the same detector response as a piece of tramp iron.

At the Caselton, Nevada, plant of the Combined Metals Reduction Co., sulphide ores are treated; therefore, it was necessary that a tramp detector operating on only the magnetic properties of the tramp iron be used.

Tramp iron detectors of this type operate on the principle that if a piece of tramp iron moves near a coil in a high intensity, steady magnetic field, the presence of the moving tramp iron causes the magnetic flux lines through the coil to change. This induces a small current in the coil. In series with the coil is generally placed a relay sensitive to currents of 1 to 3 microamp. When the tramp iron passes near the coil the flux linkages change, causing this very sensitive relay to actuate. Since the contacts of such a relay are very small it must actuate a second relay capable of breaking a larger current

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which in turn actuates another relay and finally the power circuit on the belt, thus stopping the belt.

Such a device was employed at the Pioche plant but much maintenance was necessary to keep it in operation. The very sensitive relay was not readily adjustable and was not stable in operation. It was our desire to set the device to detect rock bits but to pass small nails. It was not possible to adjust the sensitive relay to accomplish this end.

Further, as labor became more inexperienced, the amount of tramp iron increased enormously and the amount of time that the belt was stopped to remove tramp became objectionable. Most of the tramp iron can be seen on the belt if the operator knows its approximate location; therefore it was decided that instead of stopping the belt, a marker would be dropped on the belt and a warning would be simultaneously sounded to warn the operator that tramp had been detected. Then if the operator could see the tramp he could remove it without stopping the belt. If, however, he did not see the tramp the belt may be stopped and the tramp searched for in the ore pile at the point where the marker is dropped.

Certain of the steels, particularly the high manganese steels, do not exhibit very strong magnetic properties. These steels saturate in very weak magnetic fields and the presence of a large, strong magnetic field is not an aid to their detection. In a large, strong magnetic field a piece of high manganese steel will saturate long before it reaches the vicinity of the coil and while the flux linkages are somewhat distorted the rate at which they are distorted is so slow that no appreciable current is generated in the detection coil. Therefore, a field that is concentrated in a smaller area is to be desired.

Bearing these conditions in mind an electronic tramp iron detector was developed that has overcome most of the principal objections to the previous detector.

New Detector: First of all the many relays were eliminated; there is only one heavy duty relay in the apparatus that is capable of making or breaking 5 amp at 440 v.

Since the belt moves slowly, the current generated by the detector coil is essentially a direct current. It is difficult to construct a simple direct current amplifier that is stable over long periods of time. To avoid this difficulty the circuit shown in fig. 1 was used.

Between the tramp detector coil and amplifier is placed a "Brown converter," which is a special vibrating reed, single-pole doublethrow switch developed by the Brown Instrument Division of the Minneapolis-Honeywell Regulator Co. for use on their "Electronick" recorders and controllers. This device will produce an absolutely square wave 60 cycle current from a direct current source.

If current is flowing from the detector coil it is alternately connected to one end or the other of the primary of the transformer T_1. This results in an alternating current output which is fed to the alternating current amplifier consisting of a single 6SL7 GT electron tube. This tube gives a voltage gain of 3600. The output of the voltage amplifier is fed to the grid of an FG-17 thyratron. This thyratron has the heavy duty relay in its anode circuit and is supplied with anode voltage from the 250 v secondary of transformer T_3. Transformer T_2 produces bias voltage to prevent the tube from firing when there is no signal from the detector coil.

It will be noted that the grid of FG-17 thyratron is connected to the output of the voltage amplifier through a neon glow tube. This tube will not conduct unless the output from the voltage amplifier is 70 v ac at which voltage the neon glow tube breaks down, applying a high positive voltage to the grid of the thyratron. This prevents minor fluctuations in the magnetic field due to small nails, etc., which give rise to considerable short time voltages from firing the thyratron.

When the thyratron fires, contacts *a* and *b* on the heavy duty relay are made. This sounds the howler and actuates the belt marker. Also contacts *c* and *d* are broken and contacts *d* and *e* are made. When contact *d* is connected to *e* the 0.2 mfd capacitor, C_1, which has been connected across the direct current

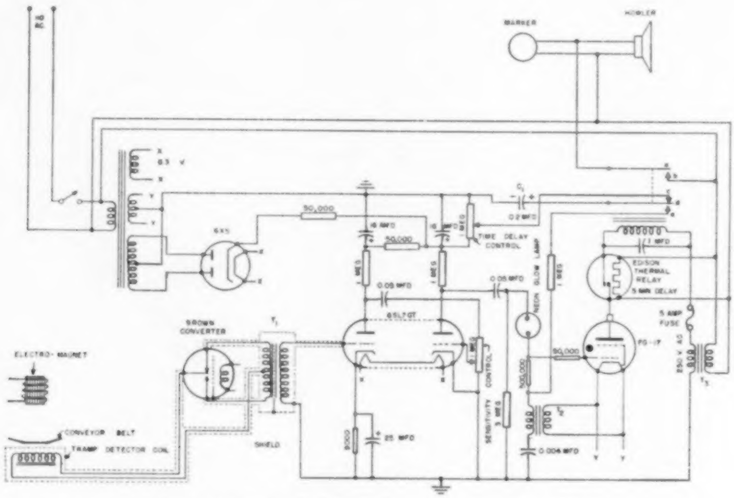

Fig. 1—Circuit of electronic tramp iron detector.

power supply of the amplifier through contact c is now connected to the grid of the thyratron. This results in a very high positive voltage being applied to the thyratron which keeps it firing for a time period that depends on the setting of the time delay control. This results in an actuation of the howler and the marker for about 1 or 2 sec, regardless of the size of the object passing over the detector coil.

In the installation at the Caselton plant the detector coil is 4 in. long and 40 in. wide and is placed under the belt and very close to it. It has 1000 turns of No. 20 wire and is completely shielded with a copper Faraday shield to prevent electrostatic pick up.

The magnets supplying the steady magnetic field are located above the belt. These are electromagnets and are supplied with direct current from a motor-generator set.

For marking the spot on the belt where the tramp iron has been detected white popped perlite is used. Other materials such as clean white sand, powdered coal, chalk, etc., may be used. The marker material is dropped in measured quantities that will make a definite white patch on the belt. The spot on the belt is never more than 2 or 3 in. out of alignment with the tramp being detected.

Through adjustment of the sensitivity control the apparatus is made sufficiently sensitive to detect iron and steel objects the size of rock bits while it does not in general detect small nails. It has been found that a 20 d nail in the bottom of the ore pile, nearest the coil, will sometimes actuate the howler and marker. It depends somewhat on the orientation of the nail relative to the coil. If it is perpendicular to the coil windings it will usually cause an actuation. When the nail is perpendicular to the winds, the most number of turns of the coil are affected by the change in magnetic field.

The Edison thermal relay is used to protect the thyratron. This tube requires a 5 min warm-up period before anode current may be drawn. The Edison thermal relay holds the anode circuit open for 5 min while the tube is warming up.

The apparatus has been in operation over two years and has required little maintenance. To date no tubes have had to be replaced.

The overall voltage gain of the amplifier and input transformer is 36,000. The apparatus is sensitive to current changes in the coil of about 1/100 microamp.

New Coil Arrangement: The principal disadvantage of the apparatus is in the arrangement of the detecting coil and magnet. The slightest movement of the coil relative to the magnet causes an actuation of the howler. To avoid this difficulty a new coil arrangement as shown in fig. 2 is being constructed. The coils and magnet will be mounted on a single rigid frame and will not be able to move relative to each other. Having the coils and magnets very close together will result in greater selectivity. Then also, there will be no dependence on an outside source of direct current power to produce the magnetic field.

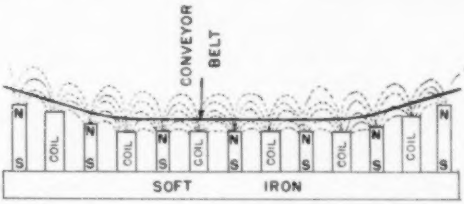

Fig. 2—New coil arrangement of detector.

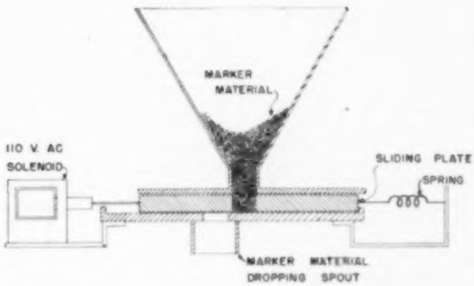

Fig. 3—Marker material hopper and dispenser.

Marker-material Hopper: Fig. 3 shows in principle the type of marker-material hopper and dispenser used. A sliding plate about ¾ in. thick with a 1 in. hole in it is normally positioned in register with a hole in the bottom of the marker-material hopper. A spring keeps the sliding plate in position. When a piece of tramp passes the detector coil the electronic mechanism causes the 110 v ac solenoid to be momentarily energized causing the sliding plate to be pulled to the left, thus putting the hole in this plate in register with the hole in the marker-material dropping spout. The marker material that fills the hole in the sliding plate is then dropped through the spout onto the moving conveyor belt.

When the actuation is complete the spring pulls the plate into position to register with the hole in the bottom of the hopper again. Bumper blocks prevent excessive travel of the sliding plate and the sharp jar at the end of the travel prevents sticking. The solenoid used has a 32 ounce pull and a travel of about 1½ in. This arrangement operates excellently when popped perlite is used for the marker material but it is not known if it would operate without sticking if another material were used.

Discussion

NORMAN WEISS:* Tramp iron detectors for protecting crushing machinery have been on the market only a few years but have already won favor with operators of crushing plants. In the crushing of sulphide ores the magnetic type of detector is most commonly used but has been criticized by some on the ground that it is difficult to regulate and is not consistent in its ability to single out only those fragments of tramp iron or steel which might damage the crusher.

* American Smelting and Refining Co., Salt Lake City.

On the other hand sponsors and supporters of the magnetic detector logically assert that it must be expected to pass through its period of trial and development just as other devices and machines have done in the past. Mr. Marquardt's paper describes a notable development which he has obtained by applying electronic tubes to increase the selectivity of the detector, and by adding a marking device to assist the operator. These changes will shorten the period of evolution.

I shall not comment on the details of Mr. Marquardt's detector, but I should like to point out that his paper expounds in considerable detail the principles of electronics while it gives only passing mention to the principles of ore crushing in which we are particularly interested. It would seem to me that Mr. Marquardt could have added value to his paper by giving us his ideas on the future field for tramp iron detectors in competition with, supplementary to, or working together with magnets. After all, a detector merely detects and manpower must do the job of removal; consequently it is only a temporary expedient, for our ultimate aim is mechanization of such operations.

I should like to mention a case in which I considered that the application of a magnetic detector would have been a mistake, simply because the actual removal of the tramp iron would have been left to the convenience and discretion of the operator. At one of our operations in Central America the ore is conveyed in lumps up to 14 to 16 in. to the primary crusher, and suspended magnets have not been able to give adequate protection to this crusher. When the installation of a detector was suggested I first thought it an excellent idea, but knowing something of the native labor I realized that they would not bother to stop the belt and remove the iron, and that if the detector automatically stopped the conveyor, the natives would simply start it again. It seemed that the only beneficial way to use the detector on this job would have been to have it stop the belt so that it could not be restarted except by the shiftboss after he had found and removed the offending piece of tramp. But this would have involved so much strenuous activity on the part of the shiftboss that the idea was thought impracticable, and was dropped.

HARLOWE HARDINGE:† Some of the control features closely parallel the circuit developments of our "Electric Ear," therefore, I have reason to believe that it can operate continuously without maintenance trouble. As I understand the circuit, it takes care of the difficult condition where materials pass the belt which should not sound an alarm, such as nails and pyrite ore. The means used to prevent a signal, through the use of a controlled voltage regulator, is certainly practical since signals below a definite voltage will not register in the alarm circuit, yet the degree of sensitivity can be controlled.

† Hardinge Co., York, Pa.

I only have one suggestion on the circuit and that is that the use of the thyratron tube FG-17 entails the use of a 5 min thermal delay relay; while if a 2050 tube were used, which cannot pass nearly as much current as the one shown in the circuit, it still should be sufficient to operate a sensitive relay of the double-pole-double-throw type to create the time delay used within the sensitive circuit and actuate the howler and marker.

Fig. 2 shows a new arrangement the author proposes to employ but this has apparently not been done as yet. In these electronic circuits, there is "Many a slip between the cup and the lip" and, therefore, until this new arrangement has actually been put in operation, it might be just as well to reserve judgment as to its practicability, although it does look as though it should work and be an improvement.

I think the means of using a marker to mark where the tramp iron is located on the belt is certainly practical and of use to the operator. I have never heard of this being done before.

Too many practical operators will consider such an arrangement as a gadget, but once they learn how to operate a device of this nature, they are going to save themselves a lot of trouble and their company quite an expense. I can see that Mr. Marquardt's comment on the sensitivity of the operation of the detecting coil and magnet is pertinent. The control at this point of all points should be rugged. It will be interesting to see how his new arrangement works.

C. M. MARQUARDT (Author's reply): The problem of removal of tramp has been enhanced by a better means of detecting the tramp even though the removal remains manual. With the marker on the belt it is a small matter to remove the tramp manually. Insofar as some better means of removing the tramp automatically is concerned I do not see much hope of accomplishing this beyond spreading the ore stream out thinner and slowing it down so the weakly magnetic tramp can be pulled from the stream with magnets, regardless of whether they be electromagnets or permanent magnets. While other means have been considered unfortunately none appear very practical or economical.

The tramp detector is best used as a means of detecting and removing tramp that gets past magnetic pulleys and surface magnets. The amount that gets by is small but it usually is the material that is most damaging to the crusher, such as detachable bits and high manganese steels.

The statement that electronic circuits are tricky must be qualified. When properly engineered and applied there is seldom anything that is less tricky than an electronic circuit. Of the many electronic devices we have constructed and are applying in our operations we have not had any difficulty with the electronic circuits. Such difficulties as have arisen have been in the mechanical devices associated with the device.

A concrete example can be given in connection with the tramp detector. On one occasion the Edison thermal time delay kept opening the anode circuit of the thyratron and thus caused failure to detect tramp. It was found that a heavy electrical load had been put on this particular circuit and the voltage was too low for safe operation of the thyratron tube. The electronic apparatus was doing just what it had been designed to do, protect itself against damage.

This apparatus has been in operation three years and to date not even a tube has had to be replaced. In the case of the previous electromechanical detector that was employed the contacts of the several relays had to be cleaned at least once or twice a month in order to keep the apparatus operating. The difference in maintenance is an ample demonstration of the dependability of the electronic circuit.

The problem of the installation of the new coil and magnetic arrangement is simply one of construction. The strength of a static magnetic field has no bearing on the problem, therefore, from an electronic point of view there is nothing tricky. Since the physics of the problem shows that the change in magnetic flux due to moving tramp will be greater than it now is, it is obvious that the vôltage pulse applied to the electronic amplifier will be greater and hence sensitivity will be greater. We anticipate that there will be some instability at very high amplification but this problem is readily corrected by merely turning down the volume control until the desired level is reached.

Progress Report on Grinding at Tennessee Copper Company

by J. F. Myers and F. M. Lewis

(With Discussion)

The paper reports the development of a large, slow speed ball mill closed circuited with a hydroscillator. This increased grinding efficiency 28 pct over conventional units.

As the title indicates, this is a progress report for the first year of experimental operation of a relatively large diameter, slow speed ball mill with 1-in. balls, with and without various classifiers, and including a hydroscillator.

Several years ago the authors came to the conclusion that if any further progress were to be made in our flotation process, an understanding of the chemical and physical reactions and the control thereof, in the grinding circuit, was essential. It may be stated, at this point, that the preparation of a correct feed for the flotation process is still the number one objective of our grinding studies. This progress report deals with grinding "per se," and not with the chemical and physical reactions as the result of grinding.

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AIME Columbus Meeting, September 1949.

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Prior to this time we had taken the naive position, that since grinding had been practiced by the industry for some fifty years or more, it had been well exploited and developed by a host of investigators. Consequently, we had given the matter only superficial attention.

At the start of our study, we were aware of the fact that some difference in opinion existed, but we were not prepared for the controversial evidence that developed nor for the extent to which the experts differed in their viewpoints. There seemed nothing else to do but to tour the continent and decide for ourselves what was right and what was wrong, read all the published literature and visit with anyone who had ideas about the matter. This we did over the next few ensuing years.

Since our study has met with some measure of success, we report, herewith, these data that ultimately they may be of help to other investigators.

In 1944 it became necessary to increase our grinding capacity and at the same time to grind the sulphide portion a little finer. This provided the impetus to put our ideas into effect. As has been previously described, our grinding flowsheet at that time consisted of a 6x12 rod mill, followed by two 5x10 ft, and one 6x12 ft trunnion overflow ball mills, with conventional rake classifiers.

Naturally, we first considered all the possibilities of altering or adding to the existing equipment. It became evident that if any appreciable increase in grinding efficiency could be obtained, it would pay to scrap the existing small mills and classifiers and install one big mill with adequate classification.

Trying to determine just how much more efficient a big mill could be, in which were incorporated all

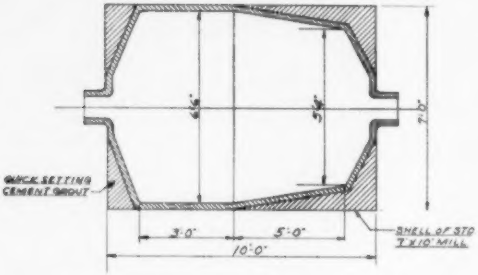

Fig. 1—Cylindrical mill converted to conical type.

the best ideas of operation, certainly brought on a lot of talk and correspondence.

Insofar as we know at this writing, there exists no rule covering the results of various size mills correlated to the other desirable operating features. It does seem, however, that grinding efficiency increases somewhat faster in a grate-type mill than in a trunnion overflow mill, as the diameter size increases. However, this statement holds true only where the grate mill is handling relatively coarse ore with relatively large balls. On fine ball mill feed, say 8 to 14 mesh, with small balls, there is a void in the record, insofar as we were able to determine, on large diameter mills.

The first stage of our study had convinced us that the rod mill was the most economical means of reducing ore from the crusher plant (¾ in. size) down to 10 or 14 mesh for the ball mill feed. By proper operation and control of the rod mill, we had eliminated any tramp oversize in the rod mill discharge, thus permitting us to go from 2 to 1 in. balls in the ball mills. This change increased the grinding efficiency 9 to 10 pct, as measured by the increase in —200 mesh material.[1] This was a gain which we did not wish to lose in going to one large mill. Our decision to use 1 in. balls eliminated consideration of a grate-type mill, as the evidence indicated that a grate mill cannot retain small balls.

We, therefore, committed ourselves to a trunnion overflow mill, of big diameter with 1 in. balls. To us there seemed to be ample evidence that slow speed offered possibilities of increased efficiency as pioneered by W. I. Garms at Hayden. Also Coghill and other investigators had demonstrated experimentally that slow speed was more efficient. We added slow speed to our specifications.

In 1945, E. H. Rose called to the attention of operators, that in a cylindrical mill, small balls tend to migrate to the feed end of a ball mill and the larger size balls migrate to the discharge end of the mill.[2] W. I. Garms proved that by correcting this horizontal ball migration, the ball mill grinding efficiency increased 6 pct.[3] This was proved by lining a cylindrical mill as shown in fig. 1. This is of course the shape of the Hardinge Tricone mill. Insofar as we know, this is the only case in the wet grinding field where the correction of the horizontal ball migration has been measured.

In the cement industry the horizonal ball migration has long been recognized as important and its correction has been accomplished by first: grate partitions in the mills; second: by Carmen liners.[4] These liners are illustrated in fig. 2. There is reason to believe that Carmen liners would work satisfactorily in a wet grinding cylindrical mill. However, this would have necessitated our designing and fitting these special liners into a cylindrical mill and it would have introduced another variable with which neither we nor anyone else was familiar, in wet grinding. The Hardinge Tricone was ready for manufacture, so our decision was to use this mill. A diagrammatic sketch of the mill is shown in fig. 3.

Mill Bearings and Motor: There seemed to be ample evidence that some 4 pct saving in power input could be obtained by carring the mill on water lubricated Micarta bearings. These were incorporated in the mill specifications. After a year of operation these have proved to be very satisfactory. It also seemed certain that our old slip-ring motors could be improved.

Classifier for Closed Circuiting: On our small ball mills we were never able to obtain a high conventional circulating sand load. There seemed to be considerable question as to just how much, if any, credit we could expect with high circulating loads when the ball mill feed was so fine. We finally concluded that we might get as much as 5 pct credit by *proper classification.*

By proper classification we had three things in mind: (1) A slime free, or nearly so, classified sand that would keep finished size material out of the ball mill and thus prevent so-called overgrinding, and the creation of reagent consuming surface in the flotation machines. (2) If the mill were going to major on lateral grinding (balls rolling or sliding on each other and not cascading), which is characteristic of slow speed mills, then it seemed to us that any lubricating effect of slimes between the small smooth balls would be detrimental. (3) From a purely theoretical angle, it did not seem improbable that by improved classification, some increase in differential grinding of sulphides and gangue could be obtained. This would decrease the power input to the ore as a whole.

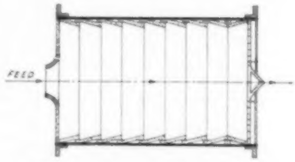

Fig. 2—Carman liner mill.

We did not know just where or how we were going to get such a classifier, as the existence of such a machine was not known. Nevertheless, we started working on the various manufacturers for such a classifier. The Dorr Co. finally offered the hydroscillator which would make a sand practically slime free. A description of this machine will not be given at this time, but will be saved for a future progress report. A general description of the Hardinge Tricone is given in table I.

Table I. Description of Hardinge Tricone Mill

Outside diameter at cylindrical section	11 ft—0 in.
Average mean diameter inside of liners	10 ft—0 in.
Rounded corners to equalize ball slippage	4 in. rad.
Discharge opening, choice of, inches	12, 22 and 33
Micarta bearings, water lubricated, inches	12 x 40
Chilled Ni-Hard liners, T.C. design	1¾ in. face
Three drive pinions, rpm	12.6, 13.7 and 15.0
Selection of percent of critical speed	51.96, 56.49 and 61.80
Mill length	9 ft—0 in.
Sychronous motor at unit power factor	500 hp
End liners of conventional ribbed type	Ni-Hard
Grinding balls "Moly Cop"	1 in. diam

Space does not permit recording the details of the preliminary information obtained regarding the value of the various power factors.

In table II we summarize in col. 1 the range of ideas of reputable engineers and manufacturers concerning these values. In col. 2, we show the values we selected in setting up the cost estimate, justifying the big mill installation. The figures shown are the percentages of power reduction required to produce a ton of —200 material.

Table II. Value of Power Reducing Factors Over Small Mill

	Preliminary Survey, Pct	Authors Estimate, Pct
Increased mill diameter	0 — 8	6
Control of lateral ball action by slow speed and correction of horizontal ball migration.	0 — 8	6
Improved motor efficiency	3 — 5	5
Micarta bearings efficiency	0 — 4	4
Power reduction by mill		21
Power reduction by classifier	0 — 5	5
Total		26

It is interesting to note that nowhere in the industry does there exist a big, slow speed mill with small balls. We realized that some degree of uncertainty existed in making such an installation.

The Tricone started operation in August 1948. In the beginning only conventional rake classifiers were available for closed circuiting the mill. These consisted of one 6-ft Dorr FX and two 6-ft Dorr Model D classifiers. By means of pumps, any combination of classifiers and mill could be made.

By October it was clear that the new ball mill, with standard classifiers was capable of equaling the old results at better than 22 pct saving in power. This can be seen in lines 12, 20, and 22 of cols. 1, 2, and 3 of table III. Since this was slightly better than the 21 pct of our estimate, it was quite gratifying.

However, the mill would pull only 308 kw at 13.7 rpm. We did not like the idea of being limited in control of the power input, especially, as we wanted to increase the capacity another 50 tons a day. Space does not permit recording the multitude of negative test results that followed, in which we attempted to make the mill draw more power.

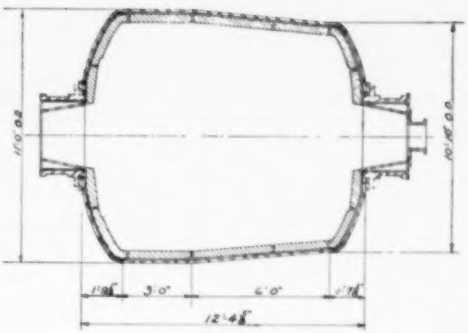

Fig. 3—Hardinge Tricone mill.

The delivery of the hydroscillator classifier was not promised until April 1949, so we could get no help from classification until then.

Among other things it was suggested that since the mill had only a 12 in. discharge opening, we might raise the ball level advantageously. We had been carrying a 45 pct volume. It is of interest to record that we added 11 tons more of balls, and the power needle did not budge.

There seemed to be nothing to do but to increase the mill speed, even though we were convinced that we would lose some efficiency. This we did (15.0 rpm, motor power 348 kw), and operated for 37 days under this setup. The results are shown in col. 4 of table III. It will be observed in line 20 and 21 that no finer grinding was obtained by the extra power and the power saving dropped to 15.3 pct (line 22).

It was evident by this time that the fluidity of the ball mass prevented the transfer of power from the shell even though the shell liners were digging into the first two layers of balls next to the shell. A check with the slow speed ball mills at Hayden did not

Table III. Operating Data

Line No.		Col. 1	Col. 2	Col. 3	Col. 4	Col. 5	Col. 6
1	Tons per day	1,900	2,050	2,100	2,100	2,000	2,100
2	Total input, kw	473	453	474	505	448	465
3	Total input, kw-hr per ton ore	5.98	5.31	5.42	5.77	5.37	5.31
4	Total input, kw-hr per ton —200	12.91	10.69	10.70	11.34	11.37	10.06
5	Total rod mill input, kw (5 ft 7 in. diam)	163	162	165	157	160	157
6	Total kw-hr per ton ore	2.07	1.90	1.90	1.80	1.92	1.79
7	Total ball mill input, kw (10 ft 0 in. diam)	310	291	308	348	288	308
8	Ball mill, rpm	27.4	13.7	13.7	15.0	13.7	13.7
9	Ball mill, pct of critical	86.4	56.5	56.5	61.8	56.5	56.5
10	Discharge opening, in.	18	12	12	12	22	33
11	Kw-hr per ton ore	3.91	3.41	3.52	3.97	3.45	3.51
12	Kw-hr per ton—200	12.34	9.54	9.68	10.45	9.95	8.80
13	Ball mill discharge, pct solids	75.0	74.3	66.9	73.0	77.2	58.0
14	Classifier overflow, pct solids	38.3	35.0	35.4	33.8	35.5	34.5
15	Rod mill feed, +0.742	5.3	2.1	2.0			
16	Rod mill feed, —200	9.8	9.6	9.7	9.5	9.5	9.7
17	Ball mill feed, +20	2.5	3.8	2.3	7.3	7.0	6.1
18	Ball mill feed, —200	24.4	23.5	23.9	22.4	22.1	22.6
19	Classifier overflow, +65	5.3	4.9	4.6	4.3	6.5	3.4
20	Classifier overflow, —200	56.1	59.2	60.3	60.4	56.8	62.5
21	Bulk concentrate, —200	65.5	65.1	64.9	65.8	57.7	73.0
22	Pct power reduction, —200 ore ball mill		22.7	22.6	15.3		28.7

Col. 1 Rod mill discharge distributed to classifiers of three small ball mills.
Col. 2 Rod mill discharge to FX, sand to Tricone and Tricone discharge to Model D classifiers.
Col. 3 Rod mill discharge direct to Tricone and Tricone discharge to Model D classifier.
Col. 4 Same as col. 2, except Tricone at 15.0 rpm.
Col. 5 Same as col. 2, except Tricone equipped with lifter bars.
Col. 6 Tricone in closed circuit with hydroscillator.

indicate that they had ever experienced any trouble by ball slippage. It will be recalled that Hayden uses rough cast balls, 2 in. in diameter.

Our liners were designed purely for protection to the shell. It was suggested that we had no wedging action from the liners to transmit power, as was the principle of the ship lap and wave type liners.

Our next test of general interest was to weld 5 in. wedge bars on to the liners at an angle of 30° to impart a wedging action to the ball mass. The results of this test are recorded in col. 5 of table III. Even after we cut the tonnage to 2000 tons per day, the results were definitely worse, and the power input reached an all time low of 288 kw. It was argued that the bar angles pocketed the balls and reduced the effective diameter. It will be noted that during the test we enlarged the 12 in. discharge opening to 22 in. (line 10) in order to observe the ball action. Our own conclusion was that 30° bars were giving a wedging action but that they reduced about 5 in. of lateral grinding from the ball mass next to the shell where it was most effective.

While the wedge bar test in itself was a failure, nevertheless, it was probably the means of our ultimate success of the project. When we opened the discharge end to 22 in., we fully expected a lot of the balls to come out, as they would have on most mills. What we saw led us to take off the rest of the trunnion plate, leaving the opening at 33 in. No balls came out, with the ball volume at 45 pct.

What we saw was a surprise to us. In fig. 4 is shown a sketch of the ball mass in action. No cascading of the balls occurs. They simply slide down the slope from the high side to level off in what one friend terms the "foot." There is no impact of balls against the down side of the shell liners or on the balls themselves, they just curl under and start over again. The action is 100 pct lateral grinding, either rolling or sliding, or both.

Insofar as we can see, the mill efficiency is just as good with the 33 in. opening as with the 12 in. opening when the mill is closed circuited with classifiers.

The Hydroscillator: The installation was made in April 1949. The laboratory tests indicated that it would increase the grinding efficiency by eliminating critical size material which is prevalent in the sand load of a conventional classifier. By critical size material we mean grains the first three meshes

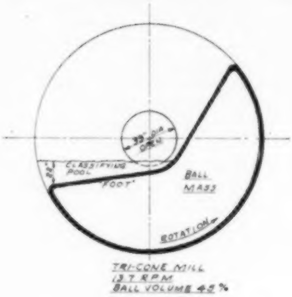

Fig. 4—Ball mill action, Tricone mill.

finer than the top size in the overflow. Aside from this, we were interested in the relative slime-free sand feature as it might minimize overgrinding and thus improve the metallurgy and reagent consumption.

The first ten weeks of varied operation with the mill closed circuited with the hydroscillator were very discouraging, but by July adjustments began to click. In col. 7 of table III is shown the effects of the hydroscillator with the mill at 13.7 rpm.

It is to be noted that the power saving on the —200 mesh ton basis increased to 28.7 pct (line 22). Under normal conditions the rake sands contain from 2 to 4 pct —200 mesh.

Classifying Pool in Mill: Referring again to fig. 4, it will be noted that over the "foot" of the ball charge there exists a classifying pool. This pool is at least 6 ft wide and 9 ft long. Since no balls were falling into the pool to disturb it, we investigated the pool characteristics.

Blocks of wood dropped into the scoop box were picked up by the scoop and deposited in the discharge box in 34 sec. Certainly any slimes or fines that got into the pool were quickly removed.

By means of vacuum lines, samples were aspirated out of the pool at various points under different conditions. From the analysis of the samples withdrawn, while operating, it is obvious that true classification takes place.

Space does not permit recording all the screen analyses of the samples taken. For illustrative pur-

Table IV. Mill Pool Samples, 12.6 rpm, Discharge 62.3 Pct Solids

	0 in. Down Left 3 ft CL*			10 in. Down 3 ft Left CL			20 in. Down 3 ft Left CL		
Screen	Pct Wt	Pct Sul	Pct Ins	Pct Wt	Pct Sul	Pct Ins	Pct Wt	Pct Sul	Pct Ins
Comp.		40.2	3.4		42.3	2.4		42.6	4.3
28	0.7			2.7	33.0	9.6	2.5	33.4	17.9
35	2.3	18.8	20.0	7.7	36.3	12.5	7.9	37.7	11.4
48	7.2	20.9	15.9	17.4	37.4	10.8	18.3	38.7	9.2
65	17.6	30.4	9.0	34.2	40.4	6.7	35.7	41.3	6.4
100	39.8	41.7	2.3	62.7	44.0	2.5	64.3	44.8	2.5
150	75.2	45.0	1.1	84.1	46.1	2.5	84.6	46.3	1.7
200	85.5	43.5	1.4	90.7	44.2	2.5	91.0	44.4	2.4
—200	14.5	38.6	2.8	9.3	38.9	4.8	9.0	38.6	4.7
Pct Solids	42.7			65.7			68.8		

* CL: center line.

Table V. Mill Pool Samples, 13.7 rpm, Discharge 47.0 Pct Solids

Screen	Ball Mill Discharge			Top at CL*			10 in. Down at CL			20 in. Down at CL		
	Pct Wt	Pct Sul	Pct Ins	Pct Wt	Pct Sul	Pct Ins	Pct Wt	Pct Sul	Pct Ins	Pct Wt	Pct Sul	Pct Ins
Comp.		35.3	10.4		36.2	10.6		38.0	8.6		38.8	7.6
28	0.3			0.4			1.0			1.3		
35	0.4			2.5	22.4	31.5	3.7	24.1	28.5	5.2	29.1	21.8
48	1.6	21.6	32.5	7.4	28.3	20.1	10.3	32.8	15.1	13.5	35.9	11.6
65	5.7	30.9	18.4	19.3	33.8	12.4	25.8	37.6	9.3	30.5	39.5	8.2
100	14.8	38.0	8.9	40.3	38.5	8.0	48.2	41.0	6.1	52.0	41.7	4.9
150	18.1	39.9	6.5	56.0	41.1	6.3	67.4	42.0	5.5	67.5	41.5	4.8
200	13.6	32.9	9.0	69.3	37.6	8.4	77.1	39.6	6.9	78.3	40.0	6.7
—200	45.5	35.1	10.7	30.7	32.6	11.9	22.9	33.9	10.4	21.7	33.6	10.7
Pct Solids	47.0			46.4			58.4			61.1		

* CL: center line.

poses of the classification effect, tables IV and V are typical of halfway back in the mill at different speeds and mill dilutions.

Summarizing all the data of the pool samples, it is evident that the rate of increase in percentage of solids with depth is rapid at the surface, but decreases continuously until at 10 in. a dilution of approximately the ultimate for the pool is reached. This would indicate that the pool is relatively quiescent in the upper zone, but turbulent next to the ball mass in the foot.

It follows that the phenomenon observed is classification and not thickening. We conclude that: (1) The rate of settling decreases as the percentage of solids increases. (2) The rate of classification decreases as the mill speed increases.

At this writing, we think the turbulence in the lower zone of the mill pool is due to pulp coming out of the ball mass and not due to agitation of the moving balls at the surface of the foot.

Summary: There remain many adjustments to make and there are many questions to answer. The effect and operation of the hydroscillator will be left to a future Progress Report; likewise, ball wear, liner wear, etc.

While we have accomplished our objective, insofar as the total power reduction is concerned, there is no evidence as yet to show just how much was contributed by each of the various factors.

Neither is it demonstrated how effective this mode of operation would be on a medium or hard ore. The Bond grindability tests show that the London ore is quite soft, namely 6.04 net g undersize per revolution at 48 mesh. Line 12 of col. 6 in table III indicates that we are producing a ton of —200 mesh material for 8.80 kw-hr as against 12.34 in the old small mills.

In the light of Bond grindability factor, the 8.80 kw-hr is not an impressive figure when compared to known accomplishments of other grinding plants in the industry. It would therefore appear that the work of the small mills must have been very poor. Insofar as we know, the small mills were doing all that was possible for them to do, and we can think of no way that their work could have been improved.

The only explanation that we have at this time is that the Bond grindability tests should have been made at 65 and 100 mesh, that possibly the individual crystals in the ore are easy to separate down to 48 mesh, but that the power required to break the crystals below 48 mesh increases materially.

Conclusion: However, we conclude, at this time, that the study of large diameter, slow speed mills with small balls and proper classification offers an interesting field of investigation. It is a subject beyond the scope of one small group of investigators with just one mill, and one hydroscillator.

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Discussion

H. HARDINGE:* The authors state that so far as they know, the test made by Garms at Hayden is the only case in the wet grinding field where correction of the horizontal ball migration or size segregation has been measured in terms of gain in grinding efficiency. In the early days of the advent of the conical mill, particularly the period from 1906 to 1916, a number of comparative tests were made and it was the result of these tests which was directly responsible for so many conical mill installations. Garms has confirmed these earlier tests by using a mill with less taper in the conical section than was used originally.

* Hardinge Co., York, Pa.

With regard to the gain due to classification, might not the 6.1 pct additional gain between col. 3 and col. 6 (22.6 to 28.7 pct) be due in part at least, to the change in pulp density and classifying effect of the pool within the mill itself rather than to cleaner sands obtained by the hydroscillator? It is noted that prior to lowering the pulp density in the mill, the results obtained with the hydroscillator were "very discouraging." It would be very interesting to see what the result would be if the mill were operated under the new conditions with a standard Model D classifier, of sufficient capacity and over a sufficient period to determine if the gain is peculiar to the combination of low pulp density in the mill with hydroscillator or if the overall result would be substantially the same in either case.

When two changes are made at the same time, it is always difficult properly to weigh the effect of each change.

Note that the authors demonstrated that a piece of wood passed through the mill in a matter of seconds. They also logically assume that slimes or fines would do likewise. Then why would not an increased amount of pulp, within reason, carrying the added slimes or fines from a less efficient classifier, pass through the mill without noticeably decreasing the grinding efficiency?

Relative Wear Rates of Various Diameter Grinding Balls in Production Mills

The results of wear on marked balls, 4, 3½, 3, and 2 in. diam are given. All balls were forged steel of practically the same chemical analysis and hardness. The results indicate that balls in a given mill for a given length of time will have equal diameter losses, regardless of size.

by D. E. Norquist and J. E. Moeller

(With Discussion)

IN order to determine the relative wear rates of several sizes of grinding balls, groups of 4, 3½, 3, and 2 in. balls were marked individually and were charged, all at the same time, into each of two production mills grinding copper ore. The remainder, and vast majority, of the ball charge consisted of 2 in. diam, and smaller, white, cast iron balls. The original weight of each test ball was determined and recorded. Some of each group of test balls were recovered from the mills periodically, individually reidentified, and reweighed. The test balls were recovered during regular maintenance shutdowns, at approximately thirty-day intervals, so as to avoid disrupting operations. As soon as the weights had been recorded, each marked ball was recharged into the mill from which it had been taken.

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AIME Columbus Meeting, September 1949.

TP 2864 B. Discussion (2 copies) may be sent to Transactions AIME before July 31, 1950. Manuscript received Oct. 10, 1949.

The test units were mechanically the same and theoretically received equal tonnages of feed.

All of the test balls were of forged, alloy steel composition with only slight variations in chemistry that would not be expected to affect the physical properties. All were heat treated in the same manner to produce a high hardness, as equal as possible for all sizes and as uniform as possible from surface to center. Generally speaking, the hardness of the portion worn from the balls during the test ranged from 62 to 65 Rockwell C. The microstructure was, therefore, predominately martensitic.

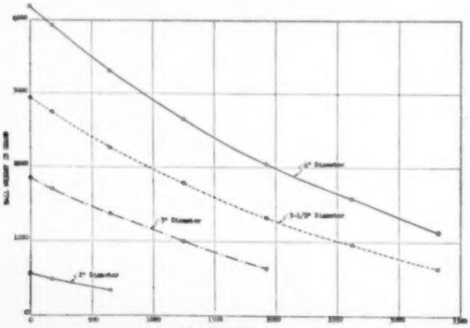

Fig. 1—Average weight loss in each ball size group in mill No. 1.

The method of marking the test balls consisted of forming a small, round hole, 3/16 or ¼ in. diam, into the center of each test ball with a Thomas Metal Master disintegrator, which produces a hole in fully hardened steel without affecting the hardness of the surrounding metal. After the hole was formed, each test ball was individually weighed to the nearest gram and the weight was recorded. The balls were grouped as to size and each group assigned a code letter. The individual balls within the various groups were then assigned a number. A small copper disc was stenciled on one side with the group letter and on the other side with the individual ball number and dropped into the hole in the corresponding test ball. The hole was then plugged with a low-melting-point metal alloy, which could be melted out in boiling water. Half of the test balls in each group were charged into one mill and the other half into another mill, so that all test balls in each mill would be subject to, as nearly as possible, identical conditions for an equal length of time.

Previous testing had indicated that the wear rates on balls of different sizes could not be accurately compared if weight loss, or percentage weight loss, were used as a basis for comparison. Fig. 1 is a graphic record of the average weight loss for each size group in mill No. 1, and it becomes readily apparent that the larger balls lose substantially more weight in a given length of time.

Fig. 2 is a graphic record of the average percentage weight loss for each size group in mill No. 1, illustrating that the smaller balls definitely lose a greater percentage of their original weight in a given length of time.

It was, therefore, decided to convert the actual weights of the test balls to diameter and follow the diameter losses periodically to see if the wear rates

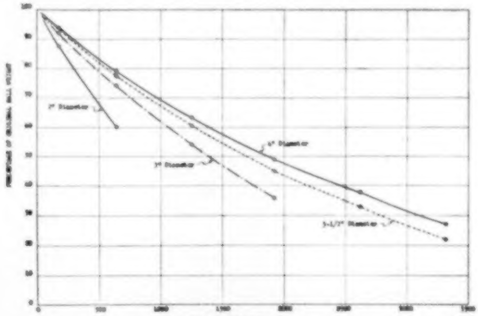

Fig. 2—Average percentage weight loss in each ball size group in mill No. 1.

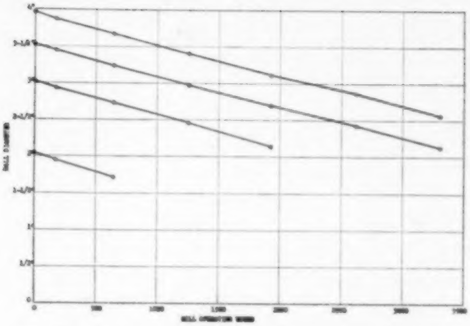

Fig. 3—Diameter loss in each ball size group in mill No. 1.

could be accurately compared on the basis of diameter loss. Table I is an example of the conversion from actual weight to diameter, and diameter loss, for each ball in one size group. The results are shown for each period that the test balls were recovered, until this group of balls had worn down to the point that a sufficient number to arrive at a reliable average could no longer be found in the time allowed for searching. A specific gravity of 7.82 was used for converting weight to diameter. Similar tables were prepared for each size group in each mill.

Table II shows the averages for each test period and the cumulative results for the various test periods on each size group in both mills.

Fig. 3 graphically illustrates the diameter loss for each size group in mill No. 1, as long as sufficient samples could be recovered representing each size group.

Fig. 4 is a graph plotted in the same manner, showing the results obtained in mill No. 2.

Table I. Original 3 in. Diam Mark N1 in Mill 1

Ball No.	Original Wt	1st Period	2nd Period	1st and 2nd Period	3rd Period	Accum. 1—3 Incl.	4th Period	Accum. 1—4 Incl.
				Weight Loss in Grams				
1	1,892	152	340	492	375	867	347	1,214
2	1,808	141	327	468	360	828	333	1,161
3	1,888	149	338	487	373	860		
4	1,886	153	337	490	373	863	352	1,215
5	1,896	153	339	492	375	867		
6	1,796	141	325	466	359	825		
7	1,896	154	342	496	374	870		
8	1,799	139	329	468	359	827	328	1,155
9	1,812	137	330	467	362	829	336	1,165
10	1,807	138	329	467				1,173
11	1,808	146	328	474	358	832	335	1,167
12	1,821	140	330	470	367	837		
13	1,892	152	338	490	377	867	347	1,214
14	1,800	141	327	468	357	825	332	1,157
Avg.	1,843	145	333	478	367	846	339	1,180
				Diameter Loss in Inches				
1	3.044	0.084	0.207	0.291	0.272	0.563	0.319	0.882
2	2.998	0.080	0.205	0.285	0.269	0.554	0.316	0.870
3	3.042	0.082	0.206	0.288	0.270	0.558		
4	3.041	0.084	0.206	0.290	0.271	0.561	0.325	0.886
5	3.047	0.084	0.206	0.290	0.271	0.561		
6	2.991	0.080	0.205	0.285	0.269	0.554		
7	3.046	0.085	0.208	0.293	0.271	0.564		
8	2.993	0.079	0.207	0.286	0.269	0.555	0.313	0.868
9	3.000	0.078	0.206	0.284	0.270	0.554	0.318	0.872
10	2.997	0.078	0.206	0.284				0.883
11	2.998	0.083	0.206	0.289	0.268	0.557	0.319	0.876
12	3.005	0.079	0.206	0.285	0.273	0.557		
13	3.044	0.084	0.206	0.290	0.273	0.563	0.319	0.882
14	2.994	0.080	0.206	0.286	0.268	0.553	0.316	0.869
Avg.	3.017	0.082	0.206	0.288	0.270	0.558	0.318	0.876

Summary

1. Diameter loss appears to be a reliable basis for comparison of wear rates on grinding balls, regardless of their original size.

2. The method described for marking and testing grinding balls in production mills is probably limited, for practicable application, to balls of about 1½ in. minimum diameter.

3. During the 3rd period, the test balls in mill No. 2, for undetermined reasons, began wearing ap-

Table II. Ball Wear for Mill Nos. 1 and 2 for Each Test Group

Period	Average Losses for Balls Recovered	Mill No. 1					Mill No. 2				
		Operating Hours	4 In.	3½ In.	3 In.	2 In.	Operating Hours	4 In.	3½ In.	3 In.	2 In.
1	Weight loss in grams		255	181	145	70		235	170	135	60
	Diameter loss in inches	176	0.082	0.074	0.082	0.088	163	0.075	0.070	0.075	0.076
	Diameter loss per 100 operating hours		0.047	0.042	0.047	0.050		0.046	0.043	0.046	0.047
2	Weight loss in grams		617	480	333	152		628	477	340	150
	Diameter loss in inches	471	0.215	0.213	0.206	0.225	467	0.218	0.215	0.207	0.216
	Diameter loss per 100 operating hours		0.046	0.045	0.044	0.048		0.047	0.046	0.044	0.046
1	Weight loss in grams		872	661	478	225		863	647	475	213
and	Diameter loss in inches	647	0.297	0.287	0.288	0.317	630	0.293	0.285	0.282	0.298
2	Diameter loss per 100 operating hours		0.046	0.044	0.045	0.049		0.047	0.045	0.045	0.047
3	Weight loss in grams		656	493	367			728	536	399	
	Diameter loss in inches	609	0.260	0.253	0.270		609	0.291	0.282	0.292	
	Diameter loss per 100 operating hours		0.043	0.042	0.044			0.048	0.046	0.048	
1	Weight loss in grams		1,528	1,154	846			1,591	1,183	874	351
through	Diameter loss in inches	1,256	0.557	0.540	0.558		1,239	0.584	0.567	0.574	0.571
3	Diameter loss per 100 operating hours		0.044	0.043	0.044			0.047	0.046	0.046	0.046
4	Weight loss in grams		613	468	339			697	510	375	
	Diameter loss in inches	668	0.285	0.288	0.318		674	0.334	0.331	0.354	
	Diameter loss per 100 operating hours		0.043	0.043	0.048			0.050	0.049	0.053	
1	Weight loss in grams		2,141	1,622	1,180			2,288	1,693	1,256	
through	Diameter loss in inches	1,924	0.842	0.827	0.876		1,913	0.918	0.897	0.928	
4	Diameter loss per 100 operating hours		0.044	0.043	0.046			0.048	0.047	0.049	
5	Weight loss in grams		459	348				479	349		
	Diameter loss in inches	697	0.254	0.263			699	0.281	0.289		
	Diameter loss per 100 operating hours		0.036	0.038				0.040	0.041		
1	Weight loss in grams		2,600	1,969				2,767	2,033		
through	Diameter loss in inches	2,621	1.096	1.091			2,612	1.200	1.185		
5	Diameter loss per 100 operating hours		0.042	0.042				0.046	0.045		
6	Weight loss in grams		446	646				442			
	Diameter loss in inches	698	0.301	0.305			698	0.317			
	Diameter loss per 100 operating hours		0.043	0.044				0.045			
1	Weight loss in grams		3,036	2,293				3,248			
through	Diameter loss in inches	3,319	1.396	1.397			3,310	1.520			
6	Diameter loss per 100 operating hours		0.042	0.042				0.046			

Note: The losses for only the balls recovered in a given period were used to determine averages; and the losses shown for the accumulated periods may not, therefore, total the same as those for the individual periods.

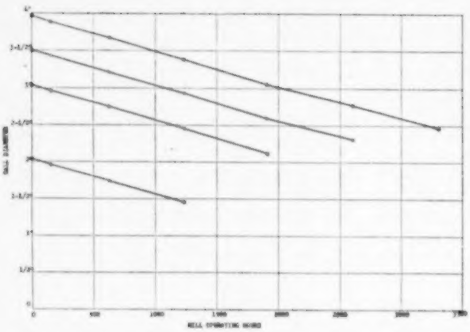

Fig. 4—Diameter loss in each ball size group in mill No. 2.

preciably faster than those in mill No. 1. The average percentage increase in wear rate for each test period was:

Test Period	Faster Wear in Mill No. 2, Pct
1	— 2.2
2	Equal
3	10.0
4	13.4
5	9.5
6	4.7

This indicates that an attempt to evaluate the quality of grinding balls by comparing the consumption of one type of ball in one mill to that of a different type in a second mill, may result in extremely misleading information.

4. A substantial reduction in wear rates occurred during the 5th period on the balls in both test mills. Subsequent investigation revealed that grinding media consumption per operating hour and per ton of ore ground decreased substantially throughout the entire plant during that month, probably due to changes in the abrasive properties of the ore.

5. The indicated life of the test balls in the test mills, assuming discharge at ½ in. diam, is approximately as follows:

Ball Size, In.	Days, Continuous Operation
2	128-133
3	213-226
3½	278-298
4	317-347

6. The length of time required to completely replace a ball charge is generally assumed to be considerably less than these figures indicate but can be quite accurately determined if properly calculated.

7. The tabulated wear rates of the alloy, forged steel, test balls are not, of course, necessarily indicative of what the wear rates would be in other mills, particularly since the test balls were mixed in with a charge of cast iron balls and the circulating load contained a quantity of cast iron grit.

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[2] T. K. Prentice: Ball Wear in Cylindrical Mills. *Jnl.* Met. and Min. Society of South Africa. (January-February 1943).

[3] F. C. Bond: Wear and Size Distribution of Grinding Balls. *Trans.* AIME (1943) **153**, 373.

[4] W. I. Garms and J. L. Stevens: Ball Wear and Functioning of Ball Load in a Fine Grinding Ball Mill. *Trans.* AIME (1946) **169**, 133: *Min. Tech.* (March 1946) TP 1984.

[5] T. E. Norman and C. M. Loeb, Jr.: Wear Tests on Grinding Balls. *Trans.* AIME (1949) **183**, 330; *Min. Tech.* (May 1948) TP 2319: *Trans.* AIME (1948) **176**, 490: *Met. Tech.* (April 1948).

Discussion

E. W. DAVIS:* The conclusions by the authors of this paper support the results presented in a paper by T. K. Prentice.† He found that the diameter loss of grinding balls appeared to be constant, irrespective of the diameter of the balls, or stated otherwise, that the rate of wear of any ball is proportional to its surface or the square of its diameter.

* Univ. of Minn., Minneapolis.

† T. K. Prentice: Ball Wear in Cylindrical Mills. Abstracted and reviewed by E. W. Davis. *Trans.* AIME (1946) **169**, 147; *Min. Tech.* (July 1944) TP 1736.

In the experiments conducted by Mr. Prentice, the mill was operated at a comparative low speed and there is some evidence to indicate that in mills operated at higher speeds the rate of wear approaches the cube of the diameter of the ball.

It would be interesting to know the per cent of critical speed at which these mills were operated and perhaps some of the other operating conditions.

T. E. NORMAN:‡ The authors are to be congratulated for the careful and painstaking work they have done on a relatively long series of wear tests. Their data provide further confirmation that grinding balls tend to lose diameter at a constant rate and thus wear (lose weight) in direct proportion to their surface area, irrespective of their weight or size. This is further evidence that the grinding forces in a ball mill are uniformly distributed over all ball surfaces.

‡ Climax Molybdenum Co., Denver.

We are pleased that the authors' findings on rate of ball wear agree with our observations[5] and also those of Prentice.[2] For a long wear test such as the authors have described in their paper, it is evident that relative wear rates should be based on the average diameter loss for each type of ball. These average diameters can be accurately determined by calculation from ball weights and densities. Where tests of shorter duration are run, we feel that the calculations can be somewhat simplified by using the factor of weight loss per unit of ball area as a basis for comparison. This factor is in direct proportion to diameter loss, so may be used provided ball areas do not change too greatly during the wear test. Most of our wear tests have involved periods of 24 to 250 hr duration. On these we have found the comparisons on the basis of weight lost per unit of ball area (area at the start of the test) to be just as accurate, within the limits of experimental error, as comparisons based on loss of diameter.

The authors have developed a method of marking which has a minimum influence on the wearing surface of each ball. This method is readily usable so long as the balls have some other distinguishing feature, such as size or soundness, which allows them to be picked out readily from the rest of the balls in the mill charge. Where, however, tests are run on balls which are similar in size and other physical characteristics with those regularly charged into the mill, then we believe it is desirable to use a more obvious distinguishing mark such as one or two notches cut in the surface of each test ball. These notches cover a greater surface area than a small drilled hole, but in spite of this they have not measurably changed the rate of wear on the groups of balls we have studied to investigate this matter.

The Colmol—A Continuous Mining Machine

by C. H. Snyder

The paper deals with details of construction of the Colmol, including improvements in design that will be incorporated in new models. These improvements are results of problems encountered and worked out in experimental operation with the unit. Also included is a descriptive plan of operating where roof and air conditions are of the worst. A summary of production results from the machine is presented.

THE various units used in conventional mining are well-built. Generally the universal undercutter, driven by approximately 50 hp, weighs approximately 25,000 lb, and is a very strong piece of machinery. The drilling machine is not as husky as an undercutter, generally is driven by approximately 15 hp, weighs about 9000 lb, but is a well-constructed, sturdy mechanism. The loader, usually 60 hp, weighs approximately 24,000 lb, and certainly is a strong, rugged unit. The working parts of all three of these sturdy conventional units have been made of tough steel alloys for some time. With the use of three machines having a combined horsepower of approximately 125, with a combined weight of approximately 60,000 lb, and with the possible inherent advantage of single purpose machines performing a single function, the use of considerable explosives has been required to complete the conventional mining cycle.

C. H. SNYDER, Member AIME, is President, Sunnyhill Coal Co., Pittsburgh, Pa.

AIME Columbus Meeting, September 1949.

TP 2865 F. Discussion (2 copies) may be sent to Transactions *AIME before July 31, 1950. Manuscript received Oct. 25, 1949.*

It was therefore our thought, upon conception of the first Colmol,* a machine which was to do the work accomplished by the three conventional units plus explosives, that it must be a rugged mechanism. Now, after building several units, after going through many and varied experiments, and after the expense of some million-odd dollars, we find we have observed and learned nothing that contradicts our original thinking. A continuous miner, if it is to give satisfactory performance, must be a rugged piece of machinery.

* Trade mark applications have been applied for, for the word "Colmol," identifying it as a continuous mining machine.

Ruggedness has been, and is constantly, in the minds of our engineers in designing the Colmol. Improvements where they have been required have been in the direction of more weight, more horse power, better alloys, bigger hydraulic pumps and motors; in short, always, toward more torque, greater strength, more ruggedness. Now not only the finest steel alloys are utilized but more than 100 Timken roller bearings are used.

The following is some pertinent data on models being built and to be built:

The low unit: Cutting height, 30 to 42 in. By changing the upper gear case the maximum cutting height can be increased to 61 in., giving a variable cutting height of from 30 to 61 in. Width of cut, 9 ft 6 in. Width of body, 75 in. over tracks. Height of body, 27½ in. Overall length, 23 ft. Driven by three 50-hp motors or a total of 150 hp. Weight, 26 tons.

The intermediate unit: Cutting height, 45½ to 72 in. Width of cut, 9 ft 8 in. Width of body, 77 in. over tracks. Height of body, 43 in. Overall length, 26 ft. Driven by three 70-hp motors or a total of 210 hp. Weight, 35 tons.

The high unit: Will have a cutting height of 64 to 88 in. Will be driven by three 90-hp motors or a total of 270 hp, and will weigh 92,000 lb.

The extra high unit: Will have a cutting height of 80 to 120 in. Will be driven by three 135-hp motors or a total of 405 hp, and will weigh 135,000 lb.

While ruggedness, weight, strength, torque are vital in a continuous miner, likewise foolproof design is important. Removing coal from the solid and loading it with one machine is a difficult task. Any machine designed to do this job, day in and day out and under greatly varying conditions, must be very powerful.

Because of the human element involved in operating the unit and because of clay veins, rock conditions, sulphur balls and similar materials and conditions existing in coal seams, which the machine may come upon without warning to the operator, some provision should be made to shut off immediately this terrific power before the unit can be damaged. Since all functions of the Colmol are performed by hydraulic or fluid drives, any sudden excessive strain or load results in the hydraulic oil going over a relief valve and causes the particular function under such load to stop. Therefore, without exerting a sudden or excessive load on electric motors or gears the function is stopped, the operator is warned, and investigation can be made. The stall or relief points are adjustable for different conditions existing in different seams. With this overload arrangement it is extremely difficult to abuse the machine. Thus, we believe we have provided not only ruggedness but comparatively foolproof ruggedness.

A continuous miner such as the Colmol, with its 3 to 5 tons or more per minute production, will obviously replace several conventional mining units. With probable daily production of 300 tons per shift in the low model and up to 800 tons per shift and more in the larger models, the mine operator will be very dependent on the production of each machine. When one Colmol is not in operation a shift's production will be greatly affected. With many mines operating two or three shifts per day, a continuous miner is likely to receive as much use in one year as many underground units now receive in two or even three years.

Such a machine must be so designed that its parts may be easily accessible and quickly replaced. With this in mind we have provided for: (1) Replacement of all bits by one man in 4 min. (This is accomplished through a novel bit holding arrangement.) (2) Replacement of hydraulic pumps and motors underground in operating height and on a unit basis. (The hydraulic pump or motor may in each instance be removed and replaced simply and quickly and without removing other gears or drives. Work may be resumed quickly and repairs may be made in the mine shop.) (3) Replacement of gear reduction units or drives on a similar unit basis.

Thus we believe we have provided for quick resumption of production and lowest possible maintenance costs.

So much for the design and construction—let us delve into the inside story of the Colmol. What have been our problems? What have we done to correct them? Where does the Colmol fit in future mining? When will machines be available?

Now, the Colmol advances into the solid on caterpillars and under its own tractive effort. In our earlier experiments we encountered considerable cat slippage. We finally resolved this difficulty with bit design. We have learned that the best bit, the one which will produce the greatest tonnage and best size of product, must provide for both cutting and then wedging or breaking the coal between it and the adjoining bit. We found that changes in wedge design could control the depth of the kerf and that depth of the kerf had a direct relation to resistance to forward movement, that is, to tractive effort required of the machine. Now, by breaking the coal between the bits, we have eliminated all track slippage and reduced hydraulic pressure required to drive the cats from its former 900 to 1200 psi to under 500 psi. However, we spent tens of thousands of dollars learning this fact.

The unit had, for a long time, a tendency to climb. It would start into the coal cutting exactly even with the bottom but would, in a matter of several feet, climb as much as several inches from the bottom. Since the kerfs between the heads on the bottom are cut out with the dozer blade and since the use of such a blade for such a purpose was apparently new, we deduced that the tendency to climb was a result of the upward resistance of these kerfs to the forward movement of the machine. After many headaches and costly experiments we finally learned, more by accident than design, that the tendency to climb was due to location of the center of weight in the body of the machine. We had long known that the center of weight including the heads (which incidentally weigh some 7 tons) was forward of center of the cats. What we did not realize was the now very obvious fact that in actual operation the greater part of the head weight was supported by the blade and the breaker heads and that therefore the actual center of weight when operating was well to the rear of the center of the cats. Thus, the back grousers were penetrating more than the front and while the machine was apparently level, it was actually inclined upward. The line of cut into the solid simply followed an incline upward parallel with the line of cat travel. This was corrected by proper computation and correction of center of weight by eliminating the proper percentage of head weight supported by the blade and heads while operating. Now by tilting the heads forward or backward or by raising or lowering the heads above or below cat level (all functions which are performed hydraulically) the operator can cut on the level, up, or down.

In the operation of the machine under actual floor and roof conditions, we encountered difficulty where rolls existed in the entry or room. Often these rolls will start up or down in the direction of the machine's travel. In many cases they will go on a line substantially perpendicular to the machine's travel. We have now provided 4 hydraulic jacks, one at each end of the cats, so that the operator may, from his operating position, raise the machine by jacking up the front or rear or both and quickly throw a timber, cap pieces or other material under front or rear or under one side to provide for a travel parallel to the coal-seam bottom.

There have been many minor difficulties. As an example, we encountered a condition of overheating in a cone drive bearing which was caused by the unusual angle of the drive and the fact that the drain was so situated that oil could not be drained from the lower end of the case and around the bearing. Sludge was permitted to accumulate at this point. A drain was quickly provided at the lowermost point of the case and the condition remedied.

Some mine men after viewing the Colmol in

operation say, "that certainly mines the coal and I can see widespread usage of the machine under good roof or where no gas is encountered, but what am I going to do where my roof conditions require cross bars within 4 or 5 ft of the face and where I have to carry line brattice up close to the face?" The following information details plans for using the Colmol which will permit timbering and carrying air up to within 4 or 5 ft of the face without undue loss of time (figs. 1 and 2).

In the alternate angle cut from center (fig. 3), it is estimated that in completing a left hand and right hand cut which we term a cycle, 29 sec are required in backing and swinging (nonproductive time) which will permit the machine to mine a 3-ft cut in 4 min. The average forward advance is computed at 18 in. per min. The amount of coal in each cycle (combined left and right hand cut) based on a 5-ft

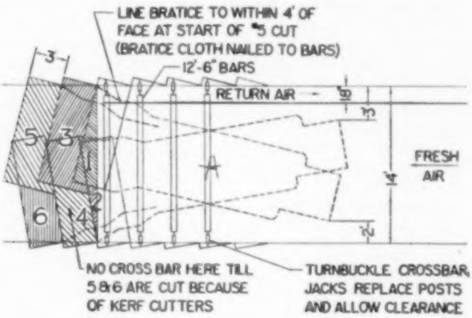

Fig. 1—Plan for bringing fresh air to face, using turnbuckle jacks and cross timbers over the machine.

seam is 12.4 tons, requiring a total mining, back up, and swing time of approximately 4½ min, which would result in a production of slightly under three tons per min, based on total elapsed time.

When one considers that the Colmol will advance a 14-ft entry or an 18-ft room a distance of 10 ft in about 10 min, an advance by conventional means requiring several hours, and when one considers that no damage is done to the roof by shooting, and still further when one realizes that the operator is back some 20 ft from face, it seems logical to assume that less timbering will be required in any given area.

Full scale production appears to be eight to twelve months away. The Jeffrey Manufacturing Co. plans to produce seven experimental Colmols of the low and intermediate sizes; the first of which will be completed in the next few days, and the last of which will be available within six to seven months. Our reason for so limiting the number is that we feel the gaining of a much wider experience in different coals and under different mining conditions is wise before full scale production is begun.

After gaining such experience with these limited number of units, a rugged brute of a machine, which will be comparatively foolproof and which will require a very minimum of down time and maintenance cost, will be offered to the industry. These units should result in extremely high production and low face costs.

Present existing intermediate transportation facilities will not permit the maximum use of the Colmol's productive capacity. Actual time studies show that under average haul, using shuttle buggies, the Colmol is able to operate only 30 to 50 pct of the time.

At present the first experimental model is operating in the Upper Freeport seam of coal at Reedsville, Preston County, West Virginia, mining an average of 50 in. of coal. The machine and a five man face crew on a 7-hr shift, when operating conditions were ideal, produced over 400 tons. During this shift the Colmol drove a single 9½-ft breakthrough of 80 ft in 90 min, producing 128 tons of coal. The daily average production to date is 225 tons per shift, which means an approximate advance of 112 ft in a standard 12-ft entry. All work being

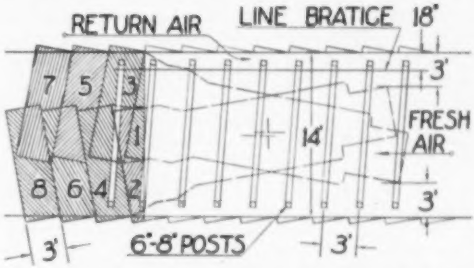

Fig. 2—Plan for bringing air to face, using 6 or 8 in. posts and cross timbers over the machine.

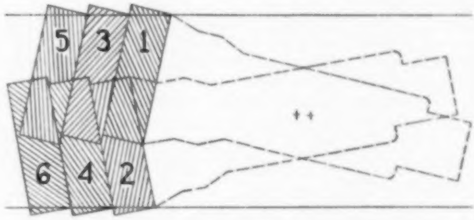

Fig. 3—Method of mining with alternate angle cut. Numerals indicate progressive steps of advance.

done at present is development work on a three entry system, where shuttle car haulage is generally over 250 ft. The delays encountered waiting for shuttle car return limit the productive time the Colmol operates in a shift. In room work this down time should be steadily decreased, permitting the Colmol more productive operating time.

Some new means must be provided for carrying the coal away from the face to the main transportation system. We are working on this problem as, we are sure, are others. We are happy to report that we have made considerable progress on paper and on models. We should soon be able to disclose a satisfactory answer to this problem—a new method which should permit some 70 to 80 pct Colmol operating time.

To gain the ultimate productive capacity with a continuous miner, new mine layouts will be desirable. During a recent visit with several of the larger operators it was indeed interesting to note that mine operators are ready and willing to consider new and different mining methods. Such an attitude on the part of mining men bids well for the ultimate success of continuous miners.

The Young Mining Engineer in the Coal Industry

by M. D. Cooper

UNDERGRADUATES in mining engineering may be prepared for work by giving them sound instruction in the courses generally considered essential to the profession. The industry is not deeply concerned about the details of those courses. The average man in the coal industry does not wish to insist upon a rigid program. Therefore, he differs little from those in the teaching profession who evidently are not unanimous in their opinions, or all college catalogs would be alike. For the good of the profession, it is just as well that there should be differences in regard to details. It appears that students graduating in mining engineering from the accredited institutions receive similar instruction.

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This paper is not a statement authorized by any organization within the coal industry, but is the concensus of opinion of a number of persons of authority in the industry and at various localities.

AIME Columbus Meeting, September 1949.

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It is taken for granted that the graduate will have a good understanding of English, mathematics, mechanics, electricity, chemistry, physics, geology, and surveying, in addition to his major courses in mining. Somewhat belatedly, industry hopes he will have had at least an introduction to the subject of labor relations, the importance of which is only too clear at present.

The coal industry expects, of course, that students in mining engineering will be taught the strictly mining subjects by men who have had practical experience in the mines and who keep themselves well informed in regard to current methods.

While the undergraduate is subject to the control of members of the teaching profession, industry expects him to be trained in certain ways that are not a part of his textbooks, but can be made an inseparable part of his development by the skillful supervision of his teachers.

Of the desired characteristics, dependability is of the utmost importance. Probably most employers would overlook certain short-comings if the young graduate demonstrated that he was thoroughly dependable. If he always appeared at the right place at the right time with the proper equipment, he would soon be well established as a welcome member of his organization. The graduate who gets a reputation for being undependable will have little opportunity for advancement.

Closely allied to dependability is loyalty. Athletic teams and social groups in college tend to develop loyalty which may well be carried over into industry. This does not mean that the graduate has to be satisfied with customary practices. The average manager is glad to see the graduate make constructive criticism as long as he demonstrates his loyalty at the same time. It is important that his loyalty keeps him alert and ready to take helpful action for the benefit of his organization, and especially to stand with it during times of stress.

With or without an introduction to labor relations in college, the graduate is expected to develop ability in this most important field. Beginning with himself, he will find it essential to deal agreeably with his immediate associates. Getting along in friendly fashion with his own small group will be a great help as his responsibilities increase and he is required to deal with larger numbers of persons. On a higher scale, his interest in his community may grow at the same time by voluntary work in any one of a great number of useful activities.

Industry expects the graduate engineer to be a mature man at the time he gets his first job. Supposing that he has better than average intelligence, industry expects him to continue to grow intellectually and to fit himself for responsible jobs when they are offered to him. For this reason, employers are apt to look over his college record to see what he did that would indicate his fitness for leadership. There is interest in knowing what he did beyond the requirements.

As evidence of his mental growth, it is expected that the graduate will do independent thinking; that he will not take too much for granted. When he reads a report, he should develop the ability to see whether the subject is new or whether it is just a description of an old method that has been superseded by something better.

For the same reason, the graduate should be able to accept conditions that have been arrived at by sound experience rather than cling to something else that seems better in theory. In this connection, it may be remarked that the ability to operate successfully a personal budget will be noteworthy, as it may be assumed that a man who knows how to conduct his own affairs will be prepared to assume the larger responsibilities of industry.

Membership in AIME will indicate to the employer that the graduate is interested in the mining industry as a whole. Therefore, it is good evidence of something more than a local outlook.

Quite apart from college training and mental ability, the newly employed graduate will be expected to be willing to do hard manual labor for a time. This will give him an understanding of the actual conditions of work done by those he supervises later. He will gain their confidence and be able to see that the work is carried on in a safe and efficient manner. Part of this experience may be acquired in his summer vacations during his undergraduate career. Such work would make a favorable impression on a prospective employer, especially if the graduate showed a willingness to continue it until he was prepared for something better.

To summarize, the man in authority in the coal industry will not quarrel with the professor of mining engineering over details of curriculum. He will be pleased if the school sends him graduates who possess a good foundation in the courses studied, and who may be depended upon to do their work faithfully and intelligently. Such men will be ready when the time comes to assume their places as leaders of an essential industry.

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aime NEWS

AIME Changes Admissions Procedures, Affecting Student and Junior Members

Changes in the handling of AIME applications will increase the efficiency of processing and reduce such expenses. Among other things, each journal will hereafter print only the names of those applicants for membership who are employed in the activities covered by the journal in question. Readers of each journal are urged to review these lists promptly as published.

Distinction between Student Chapters and Affiliated Student Societies was removed by the AIME Board of Directors. All recognized student groups are now Student Chapters, whether or not all members are Student Associates.

No further applications will be accepted for Junior Membership from anyone who has passed his 30th birthday when the application is received. Also Junior Members must change status to Associate Member or Member, and pay the $20 initiation fee (in $5 annual installments, if desired) by the time they have reached their 33rd birthday.

An exception has been made in the case of veterans of World War II who have attended college immediately following discharge from the service. They may apply for Junior Membership, even if over 30 years old, for a period of three years following receipt of a Bachelor's or professional degree, or after completion of schooling, if no degree is earned.

AIME Proposes Bylaw Amendments Regarding Members and Elections

Following the recommendation of the AIME Board of Directors, several amendments to the Bylaws of the Institute will be voted upon at the meeting of the Board on June 21.

Members—Qualifications and Elections

In Article I, Sec. 8, insert the italicized word in the second sentence as follows: "A *full-time* student in good standing. . ."

Nomination and Election of Directors and Officers

Revise the first six paragraphs of Article XI, Sec. 1, of the Bylaws as follows:

Sec. 1. A ticket of nomination for nine candidates, each to serve a three-year term as Director, of whom one shall also be designated as President-elect, and two as Vice-President, shall be prepared annually by a Nominating Committee. In 1951 the Nominating Committee shall also select, as one of the nine Directors, a President to take office in February 1952. The President-elect shall automatically become President at the end of his first year of service, and thereafter shall serve one year as Past President.

Sec. 2. The Nominating Committee, of which no member shall at the same time be on the Board, shall consist of eighteen members. Eleven of these (with an alternate for each if desired) shall be selected by the Council of Section Delegates prior to November 1. Two other members shall be chosen by the Mining Branch Council., and one each by the Metals and Petroleum Branch Councils, respectively, (each with alternates if desired) prior to November 1. The other three members, one of which shall be designated by the Board to act as chairman, shall be appointed by the President, with the approval of the Board, at the November meeting of the Board. If necessary, the President of the Institute shall also have the power to appoint a sufficient number to the Nominating Committee, at the November meeting of the Board, to bring the personnel of the committee to the authorized total of eighteen.

The names and addresses of the Nominating Committee shall be published in the December issues of the monthly journals. A meeting of the Nominating Committee shall be held during the week of the Annual Meeting at the place where that meeting is held. This committee shall proceed to the naming of a ticket. In selecting candidates the committee shall be guided by such rules of procedure as may from time to time be established by the Board, provided, however, that of the eight Directors in addition to the President-elect, five shall be apportioned on the basis of the geographical districts established by the Board. The basis of the selection of the remaining three shall be at the discretion of the committee.

The ticket thus prepared shall be transmitted to the Secretary of the Institute in time to be submitted to the Board of Directors not later than at its June meeting, and shall be published in the July issues . . . (continuing in the sixth paragraph as in the present text of Sec. 1). Sec. 2 to be renumbered as Sec. 3.

AIME Budget Set at $477,500

Prospective income for AIME, according to the 1950 budget approved by the Board of Directors at the April 21 meeting, was set at $477,500, and expenditures were set at $477,450. Two main increases in income will come from dues and advertising in the AIME's three journals. The higher rate of dues will add to revenue, but more new members are expected this year than in 1949.

AIME Revises Book Prices

Effective June 1, a new price list for AIME books will take effect. Copies may be obtained from Institute headquarters. One price will be set for each book published by the AIME. From this list price, members, Student Associates, and public libraries will be allowed a discount of 30 pct for as many copies as they may wish. Dealers will be given a 20 pct discount. Transactions volumes ordered a year in advance, at the same time dues are paid, are subject to a 50 pct discount.

Transaction volumes 1 to 183, inclusive, if in stock, are listed at $5 each. Other books will be sold at previous nonmember prices, except: "The Porphyry Coppers," $3; "Industrial Minerals and Rocks," $7; "Seventy-five Years of Progress in the Mineral Industry," $5; and "Nonferrous Rolling Practice," $4.

The Drift of Things . . . . as followed by *Edward H. Robie*

Life in the Almost Perfect State

On our last trip, we noticed this sign in a restaurant window: "Moderate Prices—Reasonably Clean Food." We still regret that we did not stop in there for a meal. In this age of exaggerations, when the tendency is always to reach up into the heights of the superlative and pull down the ultimate in language, it is indeed a relief to find such a temperate and no doubt fair statement of what is offered. It smacks of the good old days. In our hall at home we have an old clock, made by Gilbert at Winchester, Mass., and in large letters inside the case it says "Warranted Good." Not perfect, or the best ever made, but just honestly good.

Our experience has been that if something is reasonably good, or done reasonably well, one should not complain too much. We can remember back when copper ore was treated on tables and vanners and the recovery was sometimes as low as 60 pct; that was not very good but now it is well beyond 90 pct—not perfect but certainly nothing to be ashamed of and not much territory is left for future improvement in this respect. Extraction of ore and coal from mines is only reasonably good; pillars are still left. We shall never get everything out. A substantial portion of crude oil remains in the ground when a field is abandoned, but still modern recovery practices are much better than they used to be. They will never be perfect. Nor will any process ever get 100 pct recovery of high-grade gasoline from crude oil, but marvelous progress has been made. Of course, we should still strive for perfection, even knowing that it will never be attained.

Years ago one of our sons was chided for not living up to his capabilities in school, and getting "A's". "Mother," he finally said, "why can't you be satisfied with "B's" and let me be myself?"

St. Lawrence Seaway

Discovery of huge deposits of iron ore beyond the borders of the United States and current importation of some 900,000 bbl of oil per day have spurred proponents of the St. Lawrence Seaway to increased activity. Hearings have been going on before the House Public Lands Committee, with many Government officials, prominent industrialists, and farmers urging approval of the project. Until recent years United States iron ore and associated steel interests were generally against the idea, but with the present depletion of Lake Superior high-grade ores they have changed their views and now will welcome foreign ore supplies economically delivered to steel centers at and near Great Lakes ports. Congressional hearings ended on May 10. Opponents of the plan did not appear, for it has been decided not to take action at the present session of Congress.

The Seaway really has been under construction, in one way or another, for generations. Already accomplished is a channel 32½ ft deep from the Atlantic to Montreal, 1000 miles inland. A channel generally 25 ft deep is available also from Duluth to Ogdensburg, N. Y., a distance of 1250 miles. The present proposal, which dates from 1941, is to build a 120-mi canal 27 ft deep from Ogdensburg to Montreal, and to deepen to 27 ft existing waterways. The project would take seven years and cost $800,000,000. Power available for the United States would be 1,098,000 hp. Estimated traffic has risen greatly in recent years, the latest guesses being from 57 to 84 million long tons a year.

The desirability of the project is by no means universally admitted. Proponents' arguments fall into three main categories: 1—addition of needed and cheap transportation, 2—needed and cheap power, and 3—utility in case of war. Opponents have counter arguments on each of these counts. The railroads and present transportation facilities complain of the tax-free status of the seaway project and point out the loss to established facilities that would result. Further, the facilities proposed could not be used for five months of the year, and 27 ft is too shallow for 91 pct of ocean shipping. The private power interests bring forth the usual arguments against power development by the Government. They deny that there is a shortage of power in the Northeast, affirm that the proposed development would take care of only 10 pct of New York State's power needs, and say that a substantial portion of the power that it is proposed to generate could be obtained much more cheaply by further development at Niagara Falls. Opponents say that nothing is more susceptible of damage in wartime than a canal, and that any installations along our northern borders would be right out in the front trenches, so to speak, when an airborne attack comes over the North Pole.

The magazine *Civil Engineering* has published two excellent articles on the subject, one in November, 1949, favoring the project, telling something of its history, and giving details of the proposed St. Lawrence development; and another in March, 1950, marshaling the arguments of the opponents. Because of the importance of the subject to the mineral industries, AIME members would do well to apprise themselves of the pros and cons.

AMC Coal Convention Sees Industry Sound, Need for Cheap Coal by Technology

OVER 2000 coal operators and equipment manufacturers studied ways and means of producing cheaper coal to meet the tightening market at the Cincinnati Coal Convention of the American Mining Congress, April 24 to 26. Technical sessions featured roof bolting, continuous mining, and coal drying, while in the foyer "informed opinion" saw tough times ahead for some high cost producers but stabilizing influences from strong markets in power and metallurgical plants.

Senator Robertson of Virginia addressed the Monday luncheon, asserting that American constitutional liberty is being threatened by big government and by big labor. In sonorous tones he warned that constant efforts are being made to push back the boundary between government and business. Interspersing his message with choice anecdotes of the colored-Virginian-preacher type, the Senator declared that in a capitalistic system political freedom depends upon economic freedom, and that people have lost their liberties while being misled into believing they could maintain political democracy while yielding freedom of enterprise and the control over their economic destinies.

Time studies are a tool to raise the efficiency of face preparatory to operations in coal mining if the action indicated by them is carried out, remarked John K. Berry, Consolidation Coal Co., Jenkins, Ky., in his paper at the time study session. Discussing service haulage time studies, A. W. Asman, Penn State College, disclosed the results of over one hundred time studies made on different combinations of equipment. Average car change time with two shuttle cars is 1.34 min and with one car 2.66 min. With all track equipment, car change including trip change averages 2.37 min. A fair comparison of the two probably can be drawn if delays at the discharge station on the shuttle car sections are considered. The importance of this delay depends largely on total face time and will average about 6 pct of the face time.

On continuous mining, Gerald von Stroh, Bituminous Coal Research, predicted that if developments in continuous mining maintain the same rates as the past several years, cost reductions "in the order of $1 to $1.50 per ton" of coal mined by continuous machines and methods can be achieved within the next 10 years. Although experience was gained in their operation in the past year, he said, considerably more planning and attention on the part of administration and engineering personnel will be required. Operation of the Joy miner and the Colmol were described, as were the all-important subjects of service haulage and power for continuous mining.

Roof bolting is receiving a great deal of attention in both coal and metal mining. P. B. Bucky, Columbia University, described the relationship between geologic structure and the roof bolting problem as determined by theory and structural model tests. Among the advantages of roof pinning mentioned by Bucky are that it saves roof, reduces cleanup difficulties, and improves ventilation. He concluded that the classical theory has limited applications, theory is still in the development stage, barrodynamic studies are indicated, and that full scale field tests are needed.

Edward M. Thomas, U. S. Bureau of Mines, said that in 1949 approximately 200 coal mining companies were using bolts as a means of supporting about 14 million sq ft of roof surface in coal mines. "About 75 pct of these projects were initiated after April 1, when virtually all of the mechanized mines in the United States were either planning experimental installations or were preparing 'to jump head-first' into roof bolting," Thomas said. In conclusion he remarked that coal deposits heretofore considered unminable because of an extremely bad roof are now being mined safely and economically with the use of roof bolts.

The use of wooden pins for holding the roof in place at the Rio Verde mine of the Norton Coal Co. was described by Sterling S. Lanier Jr. From the beginning of the operation of this mine there was difficulty in supporting the roof and despite heavy timbering top coal and shale were falling. The company thus decided to use pins to hold up the roof. Wooden pins were tried because corrosive water would not permit the use of steel. Lanier pointed out that adoption of this method of roof control had resulted in savings in material and had reduced labor costs.

At the coal drying session, L. O. Lougee, George S. Baton & Co., said that numerous machines have been designed and the coal industry now has a choice of several makes. He said a good mechanical dryer should be able to meet these requirements: (1) ability to lower moisture content to within the limits of the customer's specifications; (2) capacity to handle and process high tonnages; (3) ability to handle large tonnages with minimum friction and degradation; (4) ability to handle large quantities of water; and, (5) provide automatic operation for the whole series of processes necessary to produce a dry product. Several types of dryers and filters to perform satisfactorily are on the market. The centrifugal dryers develop sufficient force to throw the coal and water against a fine screen through which the water passes leaving the dried coal on the inside. The process is continuous. Filters are used to dewater fine coal slurries from thickeners and froth flotation concentrates but the filtrate usually requires further drying. The filters are usually of the pressure or suction types; some are intermittent, but the continuous suction type predominates for commercial use in the United States to filter coal slurries.

Early Plans Laid for Mining-Geology-Geophysics Program at 1951 Meeting

Reported by George B. Clark

Ass't Prof. of Mining, Univ. of Illinois

FIVE topics already have been proposed for the technical sessions of the Division of Mining, Geology and Geophysics at the AIME Annual Meeting to be held in St. Louis in February, 1951. These include joint sessions of Geophysics and Geology, Mineral Education and Geology, Mining Methods and Geology, and Industrial Minerals and Geology. The occasion will also be a joint meeting with the Society of Economic Geologists.

1. Father Macelwane has proposed the topic **"Geophysics Applied to Problems in Engineering Geology."** Judging by the interest in the subject displayed at the last annual meeting, a record attendance can be expected at the two sessions planned. The first will be a series of papers and the second a round-table forum. Notification of possible contributions to these sessions should be sent to:

   Father James B. Macelwane, S.J., Dean,
   Institute of Geophysical Technology,
   St. Louis University,
   St. Louis 8, Missouri.

2. Much has been said and written in the past about the education of mining geologists and engineers, and the opinions have been almost as numerous as the speakers. It is hoped that a forum can be organized to bring this topic to more definite conclusions. Interested persons should communicate with

   James D. Forrester, Head, Mining Dept.,
   Missouri School of Mines & Metallurgy,
   Rolla, Missouri,

who is the member of the Program Committee in charge of this subject for the next annual meeting.

3. Little has been written on the geologic fundamentals which affect the choice of mining methods at various mines. To remedy this deficiency, it is hoped that four to eight papers can be secured on the subject. A good start on the problem was made at the 1950 meeting by **George B. Clarke,** Associate Professor, Dept. of Mining and Metallurgy, University of Illinois. He is in charge of this portion of the program for the next annual meeting.

4. **Roger H. McConnel,** Chief Geologist, Bunker Hill & Sullivan Mining & Concentrating Company, Kellogg, Idaho, has proposed and is in charge of developing discussion on a vital topic:

> "We all know an ore deposit when we see one. But, we don't know how to tell a weak surface or underground showing over nothing from a weak showing over a big deposit. Alteration sometimes helps; trace metal studies sometimes help; structure tied in with both of these often increases confidence. But all of these things together are none too much to convince yourself that an expenditure of several hundred thousand dollars for exploration is justified.
>
> There are literally thousands of heretofore unattractive exposures, some of which probably overlie good ore bodies. But, which ones have the best chance? This problem is something which should be discussed, more to stimulate thought than in the hope of getting distinct answers, because the problem covers the entire field of mining geology. . . . The discussion should probably be pointed to the characteristics of weakly indicated ore bodies which time has proved to be good."

5. The complexities of the clay and bauxite minerals, their occurrences, origins and treatment problems were emphasized in the last war by the demand for domestic aluminum. Research begun during the war is culminating now, and a series of sessions on these topics has been proposed by **Dr. Carl Tolman** and **Robert C. Stephenson,** Chairman, Program Committee, Industrial Minerals Division. The organization of the geologic side of this topic is under the management of **A. F. Frederickson,** Dept. of Geology and Geological Engineering, Washington University, St. Louis, Missouri; and an extraordinarily valuable session is expected.

The foregoing topics might well fill the time available for the Division of Mining, Geology and Geophysics, depending on the interest shown; *but it is also hoped individual speakers will volunteer to present papers on other topics.* Closing date for receipt of papers to be given at the annual meeting will be October 1st in order to give the committee adequate time to evaluate the papers and compose the program before the December 1st deadline. Another essential reason for the early deadline is to allow time to print the papers prior to the annual meeting.

Individual contributions on strictly **mining subjects** for the annual meeting should be addressed to:

   E. D. Gardner, Chairman, Program Committee, Mining Methods Subdivision, c/o U. S. Bureau of Mines, Washington 25, D. C.

Contributions on **Geophysical subjects** should go to:

   H. W. Straley, 3rd, Princeton, West Virginia,

who is Chairman of the Program Committee of the Geophysics Subdivision.

Geologic contributions and matters concerning the Division program in general should be addressed to:

> **John J. Collins**, Room 4203, U. S. Geological Survey, Washington, D. C.

The advice and aid of the local sections of the AIME are particularly desired in the formation of the program. It might be a helpful procedure if the local Sections would not only recommend papers for the annual meeting, but arrange for preliminary presentations on their own local programs.

Minerals Beneficiation Meets Sept. 1 at Salt Lake City

Plans are well under way for the first solo meeting of the Minerals Beneficiation Division of AIME. It will try its wings on Friday, Sept. 1 at the Hotel Utah in Salt Lake City. Though newly born, this is a healthy youngster and one can be sure of an interesting and enjoyable time at its first meeting.

Attendance is expected to be good because many mill men will be in town for the American Mining Congress' Metal Mining Convention and Exposition. The AMC show is to be held at the Fair Grounds on August 28 to 31.

The MBD will hold two technical sessions on Friday which are presently scheduled to be on crushing and grinding. Further details about the program will be forthcoming in the July and August issues. Ray Byler is program chairman and advises us that the program is taking shape rapidly.

From Salt Lake we hear from Norman Weiss, who is taking care of the on-the-spot arrangements, that the by-now famous Scotch Breakfast is on the agenda. It will be held on Thursday, Aug. 31 from 7:30 to 8:30 a.m. in the White Maple room of the Hotel Newhouse. The MBD luncheon is scheduled for Friday and there will also be a business meeting of the Division.

On Thursday, the AMC is planning some beneficiation sessions. Presently they have a symposium on heavy-media separation planned. There will also be papers on fluo-solids, and rod and ball mill liners. The exposition will be one of the biggest as over 100 booths have been contracted for. With the emphasis on mechanization and improved methods the exposition should be on the must list of every mining man.

SECTION ACTIVITIES

Ohio Valley Section

Led by a large contingent of Student Associate members assembled by Section Chairman Bill Mueller, the Ohio Valley Section, on April 21, piled into cars and spent an enjoyable and educational day as guests of the Hanna Coal Co. at St. Clairsville, Ohio.

During the afternoon the entire group of almost a hundred visited Hanna's Willow Grove underground operation, the Piney Fork washery, the Georgetown stripping operation and site of the new Georgetown washery, and the central machine shop. Highlight of the inspection trip was the chance to see in operation the world's largest stripping shovel, with a 50-yd dipper. Following one big bite, dump, and swing, the group was privileged to witness the breaking of the sheave junction on the bucket in taking the second bite, accompanied by plenty of shimmeying of the 1,800 tons of machinery comprising the entire equipment. Efficient radio communication permitted maintenance crews to start at once on the job of installing necessary replacements.

The tour was followed by dinner at the Belmont Country Club. Highlights of the Hanna Coal operations were described by James Hyslop, president, who gave a clear exposition of methods planned for the new washery to solve difficult cleaning and drying problems. Further details on coal drying, stressing results from their new Reinveld (Dutch) centrifugal drier, were given by Russell Wilmont, and on the development of the 50-yd shovel by Andrew Hyslop.

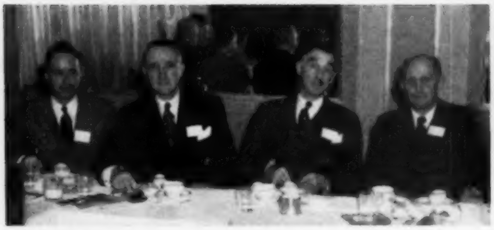
A brief business meeting concluded the program, during which the new slate of Section officers were installed, headed by Don Scott as chairman.

Southwestern New Mexico Section

P. V. Brough answered a lot of questions following his paper on cyanidation at the special technical meeting on Mar. 16 of the Southwestern New Mexico Section. Following his talk, R. C. Green, chairman, indicated that a number of young engineers, both members and nonmembers, had been invited to the meeting to discuss improvements in engineering education. Fred Coope, R. B. Tempest, Jr., Paul Smith, Don Gunther, Wesley Dow, Paul Lemke, Charles Lockhart, and Robert Shilling, were pretty much agreed that a four year engineering course was insufficient to cover the really important subjects. They seemed to feel that labor and public relations might be of more value to a young engineer than some of the courses now being taught, and that vacation experience in one's chosen field might well be a prerequisite to obtaining a degree.

Industrial Minerals Division

Phosphate Industry in Idaho, by Earl W. Murphy, started off the April 6 Pacific Northwest Industrial Minerals Division meeting of the Oregon, Columbia, and North Pacific Sections, AIME, and the Vancouver Branch, CIMM, in Roberts Hall on the University of Washington campus. Registration for the sessions began at 9, with the first



Portion of the speakers' table at the April meeting of the Washington Section, seen above includes Allen Matthews, A. Burks Summers, James Head, and J. E. Halloway of S. African Legation. Seated next to them were James Boyd, G. P. Jooste, Ambassador for the Union of South Africa, P. G. Spilsbury, and H. Dewitt Smith.

paper half an hour later. The morning session also included: Chemical Phases of Elemental Phosphorus Industry, by J. G. Miller; Technique of Identification of Nonmetallic Minerals, by J. I. Mueller; and Prevention of Hydration of Lime, by Harold Cahoon. Luncheon was served in the Student Union Building. During the afternoon, F. I. Bristol spoke on Problems of the Silica Business in the Northwest, K. E. Hamblen on Perlite in Oregon, Washington, and Idaho, and E. S. Perry on Nonmetallic Resources of Montana. The ladies joined the men for cocktails and dinner at the Meany Hotel, Seattle. On Friday there were trips to the Tacoma Smelter and the Bethlehem Steel Plant.

Washington, D. C., Section

The Union of South Africa has a progressive attitude toward foreign capital according to H. DeWitt Smith, speaking at the April 4 meeting of the Washington, D. C., Section. He told how the Union has tried to stimulate and develop its mineral resources through assisting American investment and reviewed some of the Newmont Mining Corporation's problems at the O'okiep and Tsumeb operations. The next speaker was the Honorable G. P. Jooste, who was introduced by James Boyd. The Ambassador responded in an appreciative manner to Mr. Smith's statements and assured his audience of 141 that every effort was being made by the Union to permit American capital to get an even break in mining ventures.

A. Burks Summers' kodachrome movies of a safari in East Africa, through Kenya and Tanganyika territories on a hunting expedition, brought the meeting to a close.

Southwest Texas Section

"World Wide Conservation" brought home to the members of the Southwest Texas Section at their April meeting the need to conserve our natural resources and showed the steps that petroleum engineers have taken to bring about real conservation.

William J. Murray, Jr., chairman of the Railroad Commission of Texas, speaker at the meeting, said that conservation of natural resources on a worldwide basis has become a practical necessity if the forward progress of civilization is not to be arrested and humanity forced to live on either a bare subsistence or a starvation basis. This dismal fact has been forcefully brought to the attention of mankind by numerous papers, articles, and books. It was further emphasized in papers presented by authorities from all over the world at the recent United Nations World Conservation Conference. However, this meeting also developed the equally strong conviction that there can be cause for optimism, and that the serious problems of finding and developing adequate natural resources to continue the existence and progress of civilization can be met if scientific and practical intelligence and friendly cooperation among peoples of the world are utilized. Engineers who are presumed to have the greatest combination of practical and scientific intelligence should not accept failure and ultimate catastrophe without having at least attempted to help solve that conservation problem with which they have the closest relationship.

Fort Belvoir Inspection Trip Planned for October Meeting

A visit to the research and development laboratory and an inspection of the Army Engineers' project on military bridges is being planned for Oct. 12 during the joint meeting of the Washington, D. C., Section and the Mineral Economics Division, AIME. Major General Douglas L. Weart, Com-

(Continued on page 727)

Washington, D. C., Section held its first seminar on titanium on May 3. At the speakers' table are: W. E. Wrather, C. R. Cox, a speaker, P. G. Spilsbury, chairman, E. W. Ellis, M. Macartney, O. Ralston, who gave a paper, as did W. Kroll.

At the same meeting, James Boyd, Chief Rough Neck, inducts C. R. Cox, new president of Kennecott Copper, into the Society of Hard Rock Miners. L. M. Parsons, P. G. Spilsbury, and G.F.A. Stutz enjoy proceedings.

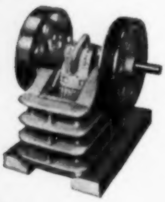
DENVER *Forced Feed* Jaw Crushers

3¼"x 4½"

$298

Price is for complete crusher with alloy iron frame, 2 flywheels for flat-belt drive. Add for grooved flywheel, V-V drive, 1 H.P. Motor and motor base. $115.

Capacity: .25 tons per hour, —¼" product.
H.P. Required: ¾.
Over-all Dimensions: (W) 16" (L) 19" (H) 17"
Specifications: Alloy iron frame, 13-14% manganese steel jaw and cheek plates (jaw plates are reversible), anti-friction bumper bearing, bronze side bearings, cast iron safety toggle.
Shipping Weight (Flat-belt drive): 365 lbs.
Shipping Wt. (V-V Drive with motor): 515 lbs.

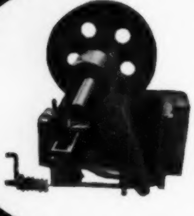


5"x 6"

$390

Price is for complete crusher with alloy iron frame, 2 flywheels for flat-belt drive. Add for grooved flywheel, V-V drive, 5 H.P. motor and motor base: $165.

Capacity: .75 tons per hour, —½" product.
H.P. Required: 3.
Over-all Dimensions: (W) 27" (L) 28" (H) 25"
Specifications: Alloy iron frame, 13-14% manganese steel jaw and cheek plates (jaw plates are reversible), anti-friction bumper bearing, bronze side bearings, cast iron safety toggle.
Shipping Weight (Flat-belt drive): 1000 lbs.
Shipping Wt. (V-V Drive with motor): 1225 lbs.

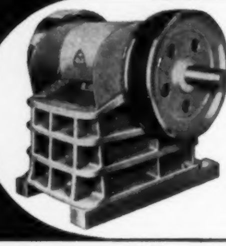


8"x 10"

$920

Price for complete crusher with alloy iron frame, 2 flywheels for flat-belt drive. (Crusher with cast steel frame $1065.) Add for grooved flywheel, V-V Drive, 10 H.P. motor and base: $272.

Capacity: 1.3 tons per hour of —½" product.
H.P. Required: 10.
Over-all Dimensions: (W) 42" (L) 45" (H) 39"
Specifications: Alloy iron or cast steel split frame, 13-14% manganese steel jaw and cheek plates (jaw plates are reversible), anti-friction bumper bearing, bronze side bearings, cast iron safety toggle.
Shipping Weight (Flat-belt drive): Alloy iron 2600. Steel 2650 lbs.
Shipping Weight (V-V Drive with motor): Alloy Iron 3010 lbs. Steel 3160 lbs.



9"x 16"

$1795

Price is for complete crusher with cast steel frame, 2 flywheels for flat-belt drive. Add for grooved flywheel, V-V drive, 15 H.P. motor and motor base: $436.

Capacity: 3 tons per hour, —½" product.
H.P. Required: 12-15.
Flywheel Speed: 250-300 R.P.M.
Over-all Dimensions: (W) 61" (L) 55" (H) 43"
Specifications: Cast Steel Frame, 13-14% manganese steel jaw and cheek plates (jaw plates are reversible), anti-friction bumper bearing, bronze side bearings, cast iron safety toggle.
Shipping Weight (Flat-belt drive): 5850 lbs.
Shipping Wt. (V-V Drive with motor): 6435 lbs.

Capacities are based on medium hard material weighing 100 lbs. per cubic foot.
8"x 10" crusher frame is cast in 2 pieces, can be sectionalized for mule-back transportation.

IN STOCK FOR QUICK DELIVERY

PRICES: Prices are firm through May, June and July 1950. Above prices are f.o.b. Denver or Colorado Springs, Colorado packed for domestic shipment only. Prices for export are slightly higher and include rugged export packing.

LARGER SIZES: Crushers up to 24"x 40" and portable crushing units are also available.

ORDER TODAY! When ordering please advise electrical characteristics. Right-hand crushers as shown will be furnished unless left-hand drive is specified. Crushers will be shipped via lowest transportation cost unless otherwise specified.

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manding General of the Engineer Center, Fort Belvoir, Va., plans to welcome members of the AIME and their guests at the Fort. Transportation by bus will be provided and luncheon will be served on the reservation. During the afternoon there will be a demonstration of engineering equipment. Headquarters for the joint meeting is the Shoreham Hotel, date, Oct. 9-13.

Coal Men Meet in June

Meeting jointly, the Central Appalachian Section and Coal Division of the AIME, and the West Virginia Mining Institute have planned a meeting for June 16 and 17 at the Daniel Boone Hotel, Charleston, W. Va.

June 16, Friday

9:00 a.m. Registration

9:30 a.m. Morning Session

"The Dreissen Cone as a Deslimer," by Donald Dahlstrom, Northwestern University.

"Application of Cyclone Thickeners to Preparation Plant Water Circuits," by H. E. Criner, Heyl & Patterson, and George Kennedy, Rochester and Pittsburgh Coal Co.

"A Comparison of Coal Sample Analyses from the Standpoint of the User," by Fred M. Reiter and Robert F. Andres, Dayton Power and Light Co.

2:00 p.m. Afternoon Session

"The Stream Pollution Problem in the Ohio Basin," by K. S. Watson, Ohio River Valley Water Sanitation Commission.

"Recent Research on Acid Mine Water Drainage," by S. A. Braley of the Mellon Institute.

"Pollution and Control of Acid Mine Drainage in Pennsylvania," by Lawrence Morgan, Pennsylvania State Sanitary Water Board.

June 17, Saturday

9:00 a.m. Morning Session

"Burning Mixtures of Virginia Anthracite and Bituminous Coal on the Detroit Rotograte Stoker," by Oscar Coplan, C. W. Kirby and C. H. Long, V.P.I.

"Primary and Secondary Mining with Auger Equipment," by Donald M. Bondurant, West Virginia University.

"The Chance Washing Process," by W. H. Lesser, Pierce Management, Inc.

Mining engineering students from the University of Washington visiting the Holden mine of the Howe Sound Co. Spring vacation means trips to properties in Washington and nearby states.

Lehigh University

Meeting for dinner at the Hanoverille Hotel, Howard Eckfeldt Society members of Lehigh played host at their April get together to Professors William Plank and Edward Martinez of Lafayette, and J. R. Ulrich and C. E. Lawall, of the Old Timers Club. Dr. Lawall presented Bob Smith, president of the society and the outstanding senior student in mining engineering, with an engraved watch from the Old Timers Club.

Dr. Gault introduced the guest speaker of the evening, G. L. Adair, of Bethlehem Steel, who spoke on magnetic surveying.

Coming Events

June 8-10, National Society of Professional Engineers, annual meeting, Hotel Statler, Boston.

June 8-10, American Institute of Chemical Engineers, Hotel Statler, Boston.

June 11-15, American Electroplaters' Society, annual meeting, Hotel Statler, Boston.

June 12-16, American Institute of Electrical Engineers, summer and Pacific General Meeting, Huntington Hotel, Pasadena, Calif.

June 14, AIME, El Paso Metals Section.

June 15, AIME, Carlsbad Potash Section.

June 16-17, AIME, Central Appalachian Section and Coal Div., and W. Va. Coal Mining Institute, Daniel Boone Hotel, Charleston, W. Va.

June 18-21, American Society of Heating & Ventilating Engineers, summer meeting, Royal Muskoka Hotel, Muskoka Lakes, Ont., Canada.

June 19-23, American Society for Engineering Education, University of Washington, Seattle, Wash.

June 19-23, American Society of Mechanical Engineers, semiannual meeting, Hotel Statler, St. Louis, Mo.

June 22, AIME, Board of Directors, New York.

June 26-27, Mining Society of Nova Scotia, Cornwallis Inn, Kentville, N. S.

July 30-31, Mining Assn. of Montana, Hotel Finlen, Butte, Mont.

Aug. 7-19, Chicago International Trade Fair, Chicago, Ill.

Aug. 28-31, American Mining Congress, metal mining convention and exposition, Fair Grounds, Salt Lake City, Utah.

Sept. 1, AIME, Minerals Beneficiation Division, Hotel Utah, Salt Lake City.

Sept. 3-8, American Chemical Society, National Chemical Exposition, Chicago Coliseum, Chicago, Ill.

Sept. 10-13, American Institute of Chemical Engineers, Minneapolis, Minn.

Sept. 15-16, National Society of Professional Engineers, Wheeling, W. Va.

Sept. 19-21, American Society of Mechanical Engineers, fall meeting, Hotel Sheraton, Worcester, Mass.

Oct. 3-5, American Institute of Electrical Engineers, Baltimore, Md.

Oct. 4-6, AIME, Petroleum Branch, New Orleans, La.

Oct. 9-13, AIME, Washington, D. C., Section and Mineral Economics Div., Shoreham Hotel, Washington, D. C.

Oct. 12-13, AIME, Southern California Section, Metal, Mining, and Petroleum Branches, Elks Club, Los Angeles, Calif.

Oct. 13, AIME, Southwestern Section, Open Hearth Committee, Iron and Steel Div., Houston, Texas.

Oct. 20, AIME, Eastern Section, Open Hearth Committee, Iron and Steel Div., fall meeting, Warwick Hotel, Philadelphia, Pa.

Oct. 20-21, Engineers' Council for Professional Development, annual meeting, Hotel Tudor Arms, Cleveland, Ohio.

Oct. 23-27, National Metal Congress and Exposition, International Amphitheater, Chicago, Ill. Participating organizations: AIME, Headquarters, Hotel Sheraton; ASM, Headquarters, Palmer House; American Welding Society, Headquarters, Hotel Sherman; Society for Non-Destructive Testing.

Oct. 27, AIME, Southern Ohio Section, Open Hearth Committee, Iron and Steel Div., annual meeting, Deshler-Wallick Hotel, Columbus, Ohio.

Nov. 9, American Mining Congress, Coal Div. Conference, William Penn Hotel, Pittsburgh, Pa.

Nov. 16-18, Geological Society of America, annual meeting, Hotel Statler, Washington, D. C.

Feb. 19-22, 1951, AIME, annual meeting, Jefferson Hotel, St. Louis, Mo. Metals Branch session to be held at the Statler Hotel.

Apr. 2-4, 1951, AIME, Open Hearth and Blast Furnace, Coke Oven and Raw Materials Conference, Iron and Steel Div., Statler Hotel, Cleveland, Ohio.

AIME Personals

1950 Quota Set at 2500 New AIME Members

"By February, 1951, 2500 new AIME members is our aim," Mr. James A. Douglas, national chairman of the membership committee, told the Board of Directors at the April meeting. Mr. Douglas is not talking through his hat when he says this because in 1949, while acting as membership chairman for the New York Local Section, he accounted for 150 new members. He accomplished this feat practically single handed and while continuing his usual routine as secretary of the Phelps Dodge Corp.

"I think this goal can be reached by the combined efforts of the Local Section membership committees, and the divisional committees with direction and coordination from our committee and the New York office," he continued. "This quota of 2500 is 15 pct of the Institute membership and is 1000 more members than joined in 1949.

"Each Local Section will be given a quota of 15 pct of its membership as a new member target. They will be requested to accept this figure or suggest a quota of their own choosing. A program of attack, which was successfully used by the New York Section in 1949, has been outlined and forwarded to the sections for their adoption if they wish. The overall campaign will be based on teamwork and personal contact in inviting individuals to become members of AIME."

James Douglas

George H. Deike, Jr.

George H. Deike, Jr., chief engineer and secretary of the Mine Safety Appliances Co., Pittsburgh, has been elected a director of the company. He has been chief engineer of the firm since 1941 and secretary since 1948.

Albert V. Applegate has taken the post of geologist with the Creole Petroleum Corp., Apt. 889, Caracas, Venezuela.

James R. Barkley has taken the job of junior engineer at the Ray Mines Division of the Kennecott Copper Co. Mail reaches him at Box 342, Ray, Ariz.

George O. Argall, Jr., has become editor of *Mining World and World Mining*, 121 2nd St., San Francisco 5.

John V. Beall has been appointed Eastern Secretary of the Mining Branch of the AIME. He will retain his position as editor of MINING ENGINEERING.

F. Bellamy has been made chief mining engineer of the Government of Pakistan, Khewra, District Jehlum, Punjab, Pakistan. He was general manager of the Singareni Collieries.

John D. Boentje, Jr., recently assumed the duties of chief engineer for the Pacific Isle Mining Co., at Hibbing, Minn., in addition to his duties as superintendent.

L. R. Brown, Jr., stopped in at AIME headquarters in April en route to his home in Rapid City, S. Dak. He is home on a four month vacation from his job with Cie. Aramayo de Mines en Bolivie, Quechisla.

Ben R. Coil has been promoted to assistant general manager of the Miami Copper Co. and the Castle Dome Copper Co. at Miami, Ariz. He had been general superintendent of the companies. **J. W. Still**, formerly mine superintendent of Miami Copper, has been made general superintendent of both companies, and **W. F. Distler**, formerly assistant mine superintendent of Miami Copper, has been promoted to mine superintendent.

Morton H. Dorenfeld has become assistant metallurgist with the Tennessee Copper Co. at Copper Hill, Tenn.

Frank S. Elfred, general manager of the Explosives Division of Olin Industries, Inc., has been elected a member of the board of directors. He has been with Olin since 1937 as a member of the executive staff.

Raymond B. Ladoo

Raymond B. Ladoo, consulting engineer of Newton, Mass., has gone to Ireland to investigate gypsum deposits recently drilled by Swedish engineers and proven to be of large extent. He will study all phases of the problem, economic as well as technologic, and try to work out a program.

John K. Gustafson has been appointed to the AEC Advisory Committee on Raw Materials. He was manager of the AEC Raw Materials Operations Office and is consulting geologist of the M. A. Hanna Co., Cleveland. **Donald H. McLaughlin** is chairman of the committee which reviews the Commission's raw materials program and advises on questions of exploration, development, and procurement.

J. S. Hazen has been elected president and director of the Hydro-Ash Corp., 1408 W. Harrison St., Chicago 7.

Vic Hollister has taken a job with the American Smelting and Refining Co. in Wallace, Idaho, as a geologist.

Robert P. Koenig has resigned as president of the Ayrshire Collieries Corp. to become president of the Cerro de Pasco Copper Corp. He succeeds **Frank F. Russell**, who becomes chairman of the board. Before joining the Electric Shovel Coal Corp., Ayrshire's predecessor, in 1936, Mr. Koenig did mining engineering and geological work in the Western States, Cuba, Bolivia, and Peru.

F. C. Lendrum became mill superintendent with the Ascot Metals Corp., Sherbrooke, Que., on April 1. He was mill superintendent with Anacon Lead Mines, Ltd.

Donald M. Liddell, secretary of the Mining and Metallurgical Society of America, and his wife arrived in Naples, Italy, on June 15 and plan to visit several Italian cities. He is looking for new technical developments in the mineral industries there.

M. B. Littlefield has joined the industrial development department of the American Smelting and Refining Co., 120 Broadway, New York City.

Charles H. Maak has become chief metallurgist with the Aerojet Engineering Corp., on the development and manufacture of rocket motors. Mail reaches him at 1621 Smith St., Ponoma, Calif.

In a recent suit involving right of way across land in a mineral area, two AIME members found themselves looking across the table at each other. **Lane Mitchell** appeared for the land owner and **Blandford Burgess** for the Power Company, in the case which was tried at Irwinton, Ga. The land had been core drilled and the testimony of these gentlemen was on whether or not the right of way included workable kaolin and glass sand deposits.

R. D. Moody, manager of Allis-Chalmers San Francisco district, has been named manager of the Los Angeles district.

Pedro Horna Moreno has been transferred to Toquepala, Casilla 57, Moquegua, Peru, care of the Northern Peru Mining and Smelting Co.

Lester R. MacLeod

Lester R. MacLeod is extending his stay in Chuquicamata, Chile, until the new sulphide plant starts operating. As representative of the N. Y. Engineering Dept., he maintains liaison and helps out the staff of the present oxide plant. With the help of Morenci, Anaconda, and Cananea experience, the plant should be the best yet. He is reached in care of Chile Exploration Co., Chuquicamata.

Charles Will Wright

Charles Will Wright, of Washington, D. C., was recently elected a director of Behre Dolbear & Co., mineral consultants, 11 Broadway, New York.

Godfrey B. O'Malley, for many years field representative of the American Cyanamid Co. in Australia, has been elected president of the Australasian Institute of Mining and Metallurgy for 1950, the highest honor which can be achieved by a mining engineer in Australia.

W. B. Plank recently was made a member of the Manpower Committee of the ASEE. This committee will report at the annual meeting of the ASEE at Seattle this month on the general engineering manpower picture.

Edmund A. Prentis, partner in the New York firm of Spencer, White and Prentis, on May 15 received the Egleston Medal awarded annually by the Alumni Assn. of Columbia's School of Engineering for distinguished service in engineering.

J. K. Richardson became industrial engineer for the western operating divisions of the Kennecott Copper Corp. on May 1. For the past four years he had been manager of the Utah Mining Association. He had worked for U. S. Potash, Climax Molybdenum, and the Tri-State Zinc and Lead Ore Producers Assn., joining the Mining Association after service in the Army during World War II.

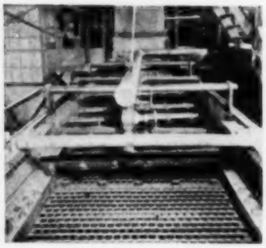
A. G. Roach, formerly ore dressing engineer with the Mining Corp. of Canada, is now mill superintendent with the Malartic Goldfields, Ltd., Malartic, Que.

J. H. Setinsky, who has been in charge of manufacturing for the Flintkote Co. for several years, has been elected a vice-president of the company. He was made general manufacturing manager in 1947 and transferred to the East Rutherford, N. J., office.

Jacob H. Sherrard, Jr., a senior in the mining engineering department at Lafayette, on Apr. 4 received a watch from the Old Timers Club, a group of prominent men in the coal industry, which cites the outstanding senior in mining engineering who intends to make the coal industry his life work and who is most likely to succeed.

Robert L. Smith, president of the Howard Eckfeldt Mining and Geological Society at Lehigh, has been awarded a watch by the Old Timers Club.

Edward H. Snyder, president of Combined Metals Reduction Co. of Salt Lake City, has been reelected for a second term as president of the American Zinc Institute, New York. **George Mixter** and **Raymond F. Orr** have been reelected vice-presidents; **Erle V. Daveler**, treas-



urer, and **Ernest V. Gent,** executive vice-president and secretary.

V. P. Sokoloff has resigned from the U. S. Geological Survey and is now with the John Hopkins University, Baltimore.

Edward M. Thomas, who has been in charge of the roof control unit set up last August in the Bureau of Mine's health and safety division at College Park, Md., is now chief of the new roof control section established at headquarters in Washington to provide increased technical advice and consultation service on roof-control problems in the nation's coal mines.

Ralph B. Utt, formerly general sales manager, has been named general manager of the Wemco Division of the Western Machinery Co. Home offices are at 760 Folsom St., San Francisco.

James Van Evera, chief engineer at the Columbia iron mine of the Inter-State Iron Co., has been transferred by the Jones & Laughlin Steel Corp. to the firm's limestone operations at Martinsburg, W. Va.

Dooley P. Wheeler, Jr., is now in charge of exploration in the western states for the American Metal Co. of Colorado, with offices at 308 Judge Bldg., Salt Lake City. He succeeds **Fred H. Stewart,** who has been transferred to Carlsbad, N. Mex., as general superintendent of the Southwest Potash Corp., subsidiary of American Metal. **Victor Bongard** has been appointed mine superintendent of Southwest Potash.

J. Ward Williams, formerly with New Goldfields of Venezuela at Guasipati, is now vice-president of Panaminas, Inc., 336 Regina Bldg., Manila, P. I.

Carl Zapffe retired as manager of iron ore properties of the Northern Pacific Railway Co. on May 1 after more than 44 years of service with the company. Mr. Zapffe is the only person living who has been identified with the Cuyuna iron range since its earliest days.

Engineers Joint Council Reports

Engineer Joint Council is composed of the presidents and secretaries of the American Society of Civil Engineers, American Institute of Mining and Metallurgical Engineers, American Society of Mechanical Engineers, American Institute of Electrical Engineers, and the American Institute of Chemical Engineers. At the year's end the total membership of the five Societies of EJC was 111,000.

A major internal step in the development of Council during the past year has been ratification by the governing bodies of the constituent societies of a formal constitution to replace the previous rather informal bylaws. The constitution states the objectives:

(a) To advance the general welfare of mankind through the available resources and creative ability of the engineering profession.

(b) To promote cooperation among the various branches of the engineering profession.

(c) To develop sound public opinion respecting national and international affairs, wherein the engineering profession can be helpful through the services of its members.

Upon that foundation, Engineers Joint Council, in its role as the cooperative, federating agency for the national societies representing the five basic branches of the profession, has been active in a wide variety of matters of national and international scope. Criteria for determining participation in any such matter are: (1) that it shall be of wide interest to the profession, and (2) that it shall be a matter having a direct impact on the profession or one regarding which it is believed that action by the profession can contribute to the public interest.

For years, EJC has been devoting

study to stimulation of greater solidarity of the engineering profession. One of the most significant steps taken during 1949 was the calling of a conference, under the general auspices of Council's Committee on Unity of the Engineering profession, for the purpose of developing specific recommendations as to measures for achieving further unity. Sixteen of the principal national engineering societies were invited to participate. At the meeting on Oct. 20 fourteen societies were represented. A committee was appointed to study the relative advantages of several suggested courses of action and to report back to the conference at a meeting to be held early in 1950.

The small expense incurred in connection with the work of EJC is significant. Each year a budget is established and the necessary funds are prorated among the constituent societies in proportion to dues collections by each for the preceding fiscal year. Expenditures may vary within rather wide limits, depending on the character of projects undertaken, but for 1948 and 1949, the total each year was about $3900.

Disbursements for 1949 were broken down as follows:

General Operating Expense and Secretarial Service.	$2498.28
General Survey Committee	697.23
Committee on Collective Bargaining	250.00
Labor Legislation Panel...	376.49
Science Legislation Panel.	73.52
Committee on International Relations	33.88
TOTAL	$3929.40

That so much can be done with so little financial outlay is a tribute to the large number of men who have contributed vastly of their time and abilities in carrying out the programs of Council, thus rendering service to their profession, to their country, and to the world. The full extent of that service cannot be appreciated except by those who have been closely in contact with the work.

A listing of the names of the active committees of EJC is shown in an accompanying box. An outline of the objectives and accomplishments of these committees is found in the EJC Annual Report, which may be had on request from the Secretary of EJC for 1950, E. H. Robie, AIME, 29 West 39th St., New York 18, N. Y.

List of EKG Committees Whose Activities For The Year Are Described in The Annual Report For 1949

Constitution and Bylaws
General Survey; Labor Legislation
Increased Unity in the Engineering Profession
Science Legislation
Engineers Cooperating in Medical Research
Engineers in Civil Service
Selective Service; Fuel Resources
Federal Income Tax
National Water Policy
National Engineers; UNESCO
International Relations
Consultative Status with the United Nations

Obituaries

Tom Lee Ball (Member 1943), former director of the division of safety and inspection of the State of Alabama, died June 24, 1949. Mr. Ball was a native of Florence, Wisc., born there in 1888 and was graduated from Washington and Lee. For many years he was superintendent for the Gulf States Steel Co. and in 1942 joined the Alabama By-Products Corp., leasing coal lands and superintending the Colta mine.

Eugene Henry Dawson

AN APPRECIATION BY HARRY J. WOLF

Gene Dawson (Member 1919), mining engineer and world traveler, died Mar. 14 of a heart attack following a game of chess, which he won, at the Mining Club in New York. He was born in Bloomington, Del., in 1879, and spent most of his early life in Colorado, graduating from the University in 1905. He was associated with the Mining Exploration Co. in Central and South America; with A. Chester Beatty in South America and Canada. In 1918 he was engaged on special work for the Materials Production Department of the Signal Corps, then became general manager for the Fairhaven Water & Mining Co. in Alaska, where he developed the method of thawing frozen gravel with cold water. He was with the Whitworth Finance Co. in London, Consolidated Goldfields in the Yukon, Selection Trust in Norway, Mines Development Ltd. in Yugoslavia, the states of Kashmir and Bopal in India, Sindicate Coloniale in Italy and Africa, Rogers, Mayer & Ball in New York, the WPB in Washington, and Behre, Dolbear & Co. in New York.

Eugene Dawson was endowed with sound judgment, absolute integrity, and a good sense of humor. Whenever anyone related an anecdote, Gene could be counted on for a "that reminds me" which would be a judicious mixture of novelty and humor. Despite his wide experience, he was not in the slightest degree conceited, and never sought publicity. To his friends he was good natured and jovial. Close associates enjoyed his assumption of an artificial belligerency in an argument, especially when he had a definite opinion to express—and that is the kind of opinion he usually had. His sudden departure was a shock. He will be missed and long remembered.

John Blakeney DeMille (Member 1934), died recently in Italy while on an ECA mission to study that country's mineral resources. He was 50 years old. Mr. DeMille, after graduating from Columbia, worked for the N. Y. State Geological Survey, the American Hydrocarbon Co., and made examinations and appraisals in eastern Canada and the States. In 1932 he had established offices in Montreal as a consulting geologist and mining engineer, examining, appraising and developing mining properties. He was in on the discovery of the iron deposits in Labrador in 1936.

Gideon Boericke (Member 1919), owner of the Primos Chemical Co., died Mar. 19. He went to work for Stein & Boericke, Ltd., after graduating from Lafayette, designing coal washing plants and coke ovens. In 1901 he became treasurer of Primos Chemical Co., mining tungsten, molybdenum, and vanadium, and in 1919 was made president of the company, carrying out research on those metals.

Grant H. Dowell (Member 1916), who celebrated his 83d birday last September, died Mar. 29 in his San Marino, Calif., home. Mr. Dowell had worked for Phelps Dodge, Old Dominion Co., Copper Queen Consolidated Mining Co. He retired from Phelps Dodge in 1927 because of ill health, but acted as a consultant for the company when he could.

Clarence N. Fenner (Member 1896), research geologist of Clifton, N. J., died Dec. 24, 1949. He was 79. Mr. Fenner was graduated from Columbia in 1892, did work in Central and South America, and by 1912 was doing scientific research at the Geophysical Laboratory of Carnegie Institution in Washington, remaining there until the late 1930s, when he moved to Clifton.

Robert Lanbach Klotz (Associate Member 1935), formerly with the Hercules Powder Co., is dead. He was born in 1881 and studied at Lafayette. For nine years he was with E. I. du Pont de Nemours Powder Co. and in 1913 became sales manager for Hercules Powder.

Henry Marquette Lane (Member 1899), consulting engineer of Grosse Ile, Mich., died Aug. 8, 1945. As a boy of 12 he attended his first AIME meeting with his father. For many years he was president of the H. M. Lane Co., builders of industrial plants, but specializing in foundries and foundry problems here and in Europe.

Leonard Larson (Member 1918), general superintendent of the reduction plant of the Kennecott Copper Co. at McGill, died Feb. 12, 1950. He had been in poor health for several years. Although he went along with his parents to Wyoming when a youngster, Mr. Larson was born at Roy, Utah, in 1886 and graduated from the University of Utah in 1911. Right after graduation he went to work for the Kennecott Copper Co. as an experiment man and assistant control chemist. As his know-how increased he became roaster engineer, roaster general foreman, assistant smelter superintendent, superintendent of the smelter, and in 1930 was made general superintendent.

George Dietrich Nordenholt (Member 1914), consulting mining and petroleum engineer of Beverly Hills, Calif., died May 23, 1949. Mr. Nordenholt was born in 1884 and graduated from Colorado School of Mines in 1909. He worked for the Los Angeles Aqueduct, the Domingo Mines, Pacific Mines, and the Idaho Mines Corp. He was president of the Natural Gasoline Co.

Herbert T. Quin (Member 1942), engineer with the Philadelphia & Reading Coal & Iron Co., died June 14, 1949. After graduating from Lehigh, he worked briefly for the Pittsburgh Crucible Steel Co., going to the Lehigh Valley Coal Co. in 1914. Service in the Field Artillery interrupted his career; upon return to civilian life he joined Philadelphia & Reading Coal & Iron.

Perry Curtis Robbins (Member 1945), division engineer for the Koppers Coal division of the Eastern Gas & Fuel Associates, died Feb. 19, 1950. He was 58 years old. Mr. Robbins joined Koppers Coal in 1924 as resident mining engineer. He had been with the Stonega Coal & Coke Co., Hillman Coal & Coke Co., and G. M. Jones Coal Co.

Peter Marseilles Saxman (Member 1916), president of the Saxman Coal & Coke Co., died April 27, 1949. He had been president of the com-

Necrology

Date Elected	Name	Date of Death
1936	Charles B. Andrews	Apr. 20, 1950
1900	Frank Ashton	Unknown
1921	George Belchic	Sept. 3, 1949
1919	Gideon Boericke	Mar. 19, 1950
1929	Walter W. Bradley	Apr. 11, 1950
1944	Donald Wynn Crosby	Nov. 29, 1949
1916	Grant H. Dowell	Mar. 29, 1950
1914	Francis G. Fabian	Dec. 13, 1949
1896	Clarence N. Fenner	Dec. 24, 1949
1949	Edwin S. Giles	Mar. 8, 1950
1923	Charles S. Hurter	Jan. 2, 1950
1946	Joseph H. Jones	Apr. 18, 1950
1925	William A. Kissam	Jan. 1950
1943	Arthur S. Knoizen	Apr. 29, 1950
1948	David A. Park	Apr. 13, 1949
1918	James G. Parmelee	Mar. 1950
1937	John P. Shannon	Dec. 28, 1949
1927	Ladislaus Szily	Feb. 20, 1950
1942	John P. Wilcox	Feb. 7, 1950
1914	Oba Wiser	Feb. 4, 1950

pany since 1930 and had worked for the company in various capacities previous to that. Mr. Saxman had served with the Central Iron and Steel Co., the Ebensburg Coal Co., and Electro Metals Ltd.

William Willard Taylor (Member 1897), mining and chemical engineer of Signal Mountain, Tenn., is dead. Mr. Taylor was graduated from the University of Michigan in 1893 at the age of 22. He worked for the Illinois Steel Co., Missouri Furnace Co., Empire Steel & Iron Co.; he was general superintendent of the Allegheny Ore & Iron Co., vice-president of the Victoria Coal & Coke Co., vice-president and general manager of the Oriskany Ore & Iron Co., and the Chattanooga Iron & Coal Co.

Nicolas N. Vishnevsky (Associate Member 1938), formerly with the American Smelting and Refining Co., died in Salt Lake City on Mar. 19. He was born in Russia in 1888, worked for the government, became an officer in artillery, and in 1916 was sent to Washington for the government. He remained in the States with the Bureau Technique Francais and in 1922 joined the Utah Copper Co. He also worked for the Clayton Mining Co. and in 1936 began his association with American Smelting.

Richard Alexander Wagstaff (Member 1927), inventor of the C-W roaster and Wagstaff gunfeed for reverberatory furnaces, died Mar. 9 at his home in Salt Lake City. He was 62. Mr. Wagstaff had worked for Utah Copper Co., Butte-Duluth Mining Co., and Ray Consolidated Mining Co. before he joined the Garfield smelter of AS&R in 1915. In 1936 he became assistant to the general manager of all western department operations and vice-president. He was consultant for Nevada Consolidated and Phelps Dodge. He also wrote books on mining and metallurgy.

Herbert Allen Wheeler (Member 1881) passed away at the age of 91 on Mar. 11 at his home in Webster Groves, Mo. A member of the AIME for almost 70 years, Mr. Wheeler has long been listed among the members of the Legion of Honor and was an active member of the "Faithful" in the St. Louis Section. Following his graduation from Columbia in 1880 he went to Utah to work for the USGS and the DRG&W Railroad Co., returning East in 1882 to superintend the Ely Copper Mine in Vermont. He was a member of the faculty of Washington University in St. Louis, and manager of the Standard Tile Co. From 1900 to 1947 he was a consulting ceramic and mining engineer and oil geologist with offices in St. Louis, retiring in 1947.

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on Mar. 31, 1950, was 16,397; in addition 4037 Student Associates were enrolled.

ADMISSIONS COMMITTEE

E. C. Meagher, Chairman; Albert J. Phillips, Vice-Chairman; George B. Corless, Lloyd C. Gibson, Ivan A. Given, P. Malozemoff, Richard D. Mollison, and John Sherman.

Institute members are urged to review this list as soon as the issue is received and immediately to write the Secretary's Office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate.

Alabama
Birmingham—Fanning, Alfred N. (C/S—S-J). Maples, George R., Jr. (M).

Arizona
Benson—Sganga, Frank M. (C/S—S-J).
Ray—Ebert, William H. (C/S—S-J).
Tucson—Ruff, Arthur W. (C/S—S-J).

Connecticut
Old Greenwich—Hearon, Henry H. (AM).

Colorado
Colorado Springs—Newton, Douglas E. (C/S—S-J).
Denver—Hollister, John C. (M).
Jamestown—Ovitz, Ernest G. (R, C/S—S-M).

District of Columbia
Washington—Bauman, Edward W. (M).

Florida
Arlington—Carpenter, James H. (M).

Idaho
Kellogg—Coulter, William J., Jr. (R,C/S—S-J).
Weiser—Lovejoy, Victor E. (C/S—S-J).

Illinois
Carlinville—Moran, John T. (C/S—S-J).
Urbana—Silverman, Maxwell. (J).

Kentucky
Drift—Reed, Boyd F. (AM).

Minnesota
Coleraine—Pitmon, Guy R. (C/S—S-J).

Missouri
Bonne Terre—DeClue, Benjamin F. (M).
St. Louis—Robertson, Florence. (M).
Webster Groves—Heinrich, Ross R. (M).

Montana
Butte—Bauman, Harry C. (C/S—S-J).

New Jersey
Livingston—James, Ulysses S. (M).
Rockaway—Speal, Alexander J. (C/S—S-J).

New Mexico
Carlsbad—Libbey, Donald L. (C/S—S-J).
Hurley—Gunther, Donald R. (J). Jacobs, Bernard C. (R,C/S—J-M). Hurley, Paul A. (R,C/S—J-M).
Santa Rita—Tempest, Rone B., Jr. (R, C/S—S-M).

New York
Bronx—Brown, Joseph F. (C/S—S-J).
Douglaston—Stehle, Frederick T. (J).
Harriman—Mulligan, John J. (C/S—S-J).

Appraisals
Assayers
Chemists
Construction
Consulting
Designing

Professional Services

Space limited to AIME members or to companies that have at least one member on their staffs. One inch, $40 per year; half inch, $25 payable in advance.

Oakfield—Olsen, Lyle M. (C/S—S-J).

Ohio
Westerville—Gribble, Robert Q. (J).

Pennsylvania
Carbondale—DeAngelis, Joseph T. (AM).
Lansford—Fauzio, Frank Pat. (M).
McKeesport—Frazier, Samuel B. (J).
Nanticoke—Christian, William E. (M).
State College—Hoffman, John N. (C/S—J-M).

South Dakota
Custer—Norton, James J. (C/S—J-M).
Rapid City—Hammerquist, Donald W. (C/S—S-J).

Texas
Houston—Clark, George H. (M).

Virginia
Richlands—Bagenstose, Raymond M. (M).

Alaska
Anchorage—Anderson, Ludlow G. (R,C/S—JA-M). Boyle, Thomas L. (M). Chambers, Glen D. (AM).
Spenard—Graham, Francis C. (M).

Quebec
Asbestos—Borror, Charles D., Jr. (J).

Brazil
Rio de Janeiro—Pimenta, Dermeval J. (M).

Peru
Julcani—Conroy, Milton R. (M).

Southwest Africa
Tsumeb—Griffis, Arthur T. (R,C/S—S-M).

Turkey
Istambul—Kromer, Ferid H. (C/S—A-M).

Advertisers' Index

NEW PUBLISHING SCHEDULE

SPACE RESERVATION — 1st PRECEDING MONTH
PLATES IN NEW YORK — 7th " "
MAGAZINE ISSUED — 23rd " "